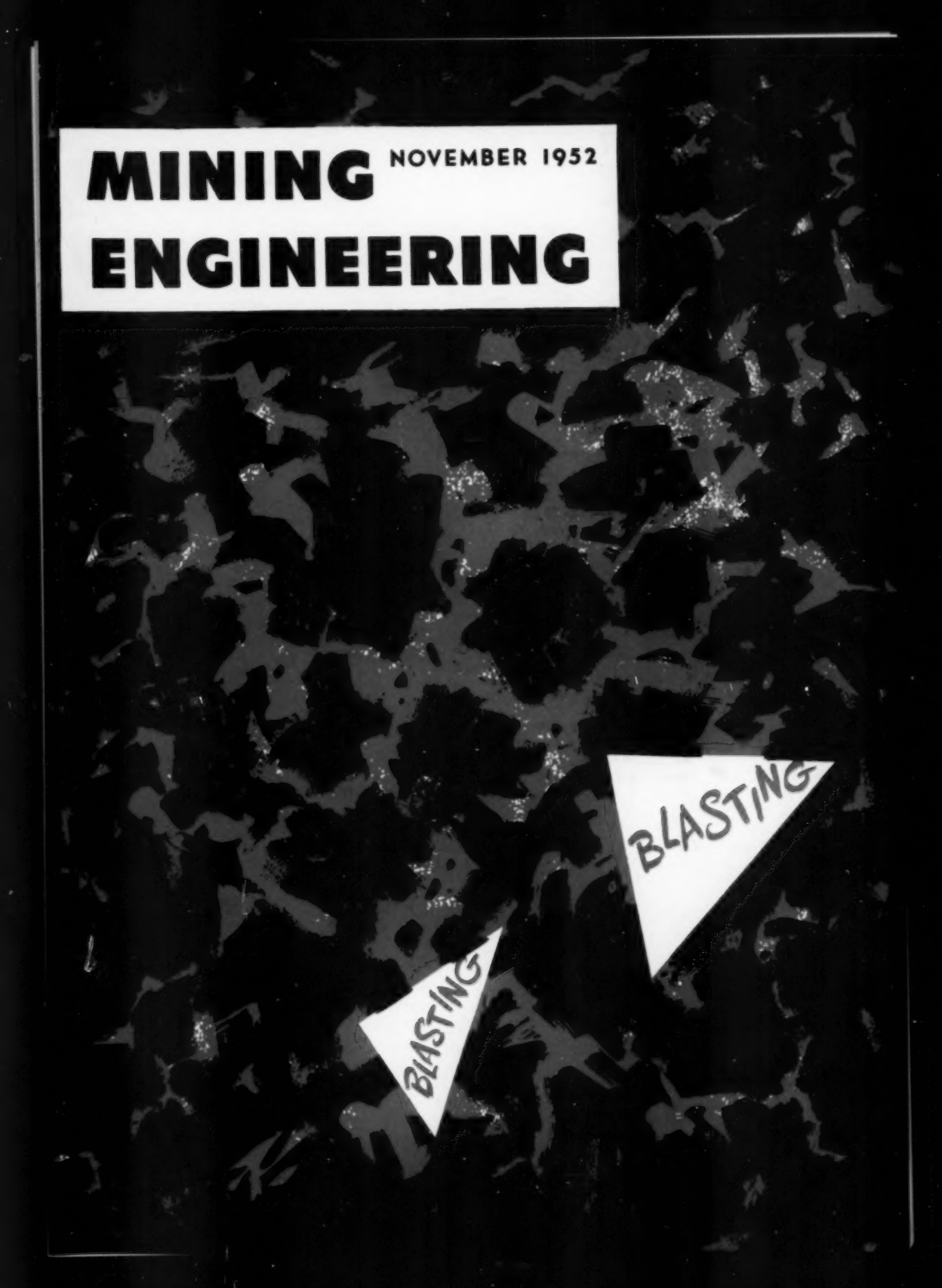
MINING ENGINEERING

NOVEMBER 1952



WILFLEY
centrifugal PUMPS

30 acres 2 miles

• ***Wilfley Special High Head Sand Pumps*** play an important role in the successful reclamation program of the Hawaiian Commercial & Sugar Co.

• Each year, at the Puunene Mill, 30 to 40 acres of useless, sandy waste land are transformed into productive ground for sugar cane cultivation.

• Soil, removed from mechanically harvested cane by washers, is settled to a sludge of 15% to 25% solids. This sludge, together with a sludge of repulped filter cake, is pumped to the diked waste area. When dry, there remains a 2 to 3-ft. layer of reclaimed cultivable topsoil over the coral sands.

• WILFLEY Pumps, operating on a 24 hour-a-day, trouble-free schedule perform the tremendous task of delivering 300- 500 G.P.M. of cane washer wastes through 2 miles of 6" pipeline to the reclamation area. As a result, wastes which formerly presented a costly disposal problem, are now converted into revenue-producing land

Wilfley Special High Head Pumps in operation at the Puunene Mill of the Hawaiian Commercial & Sugar Co.

Buy rugged, heavy-duty Wilfley Pumps whenever the job calls for the efficient and economical handling of sludges, slimes, slurries or sands. The new WILFLEY Model "K" Centrifugal Sand Pump, embodying many important mechanical improvements, delivers higher efficiencies, stepped-up production, worthwhile power savings and lower operating costs. Easy interchangeability of wear parts. Low maintenance costs. Every application individually engineered. Write or wire for complete details.

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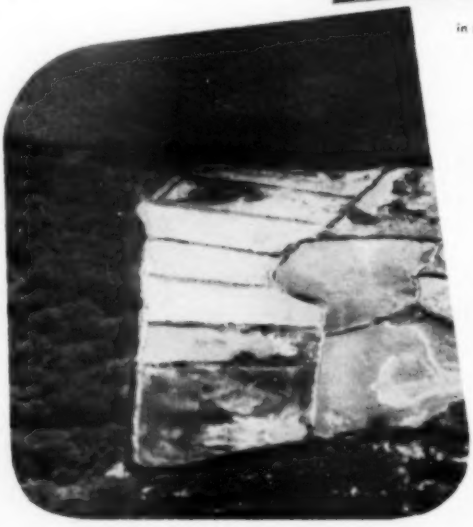

Buy WILFLEY for cost saving performance

Companion to the famous WILFLEY Acid Pump

Upper: Reclaimed land under cultivation.
Center: Diked reclamation area.
Bottom: Waste land to be reclaimed.

MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 4 NO. 11 NOVEMBER, 1952

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COVER

This month's cover, by artist Herbert McClure, is based on pictures taken of a blast with a new machine gun type still camera. The photographs and their interpretation will be found on page 1041.

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— Personnel Service —

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Engineering Manager, many years' experience in design, construction and operation of industrial plants and mining properties, requiring a knowledge of mechanical, electrical, civil and mining engineering. Have directed many men and engineers working on foregoing projects. Domestic and foreign experience. Now available. M-702.

Mining Engineer, 36, married. Thirteen years' experience underground mining operations; exploration, development, mine evaluation, cost estimation and job evaluation. Registered Professional Engineer. Employed. Available reasonable notice. M-697.

Mining Engineer, B.S. plus 2 years' pre-law. Three years' experience; completed training program all phase open pit and underground. Last 2 years sintering, nodulizing and wash plants. Varied pre-professional job experience. Veteran, 32, married, 2 children. Available immediately. Will consider any location. M-703.

Experienced Mining Engineer, 34, single; Master's degree; Registered Professional Engineer. Sixteen years' operation, engineering, examination, management experience in United States, Alaska and Canada. Mostly hardrock, but some placer experience. Available short notice. Prefer western United States or Canada. M-704-4710-E-5-San Francisco.

POSITIONS OPEN

Mining Engineer for consulting work. Familiar with modern methods of surveying and mineral extraction or ore mining methods as well as mining machinery requirements within the limits of actual need. Must be able to do occasional surveying on the job. Salary, $11,000 with possibility of becoming chief consultant. Location, upstate New York. Y7643.

Mining Engineer for limestone mine, young, graduate, with several years' practical experience, preferably in the operation of a draft operation mine. Must have supervisory ability. Company engaged in a modernization program involving the construction of a new three thousand barrel per day cement plant. Will put into practice the recommended methods for new and more efficient mining techniques, and will supervise future operations. Location, Missouri. Y7609.

Engineers. (a) Chief Engineer, graduate, mining, who has had 5 to 10 years' experience in surface and underground surveying, drafting and ore reserve calculation planning of surface and underground installations. Two year contract. Location, Central America. (b) Jr. Geologist, who has had some field experience, either during summer vacations or 1 year after graduation, for large mining company. Salary, $3600 a year to start. Location, Central America. Y7589.

Ore Geologist or Mining Engineer with experience in surveying primary and alluvial deposits to take charge of an expedition making a survey of tin ore deposits. Salary open. Location, South America. Y7231.

Mill Superintendent to take charge of a 100-ton flotation gold mill. Must speak Spanish. Elevation, 9000 ft. Salary open. Location, Argentina. Y7216.

Chief Engineer, mining graduate preferred, but will consider civil or mechanical with about ten years' experience, to include either construction or underground operations. Mining is open-pit but transportation problem is involved so should have some experience in mechanized operation. Transportation paid; housing available; salary open. Location, West Indies. Y7180.

Mining Engineer with some engineering training, capable of doing mapping, surveying, exploration, geology and transit work. Location, New York State. Y7333.

Research Engineer, 25 to 40, graduate in mining or metallurgical engineering, preferably with a Master's degree in mineral dressing, with at least three years' experience in ore dressing, research or mill work, preferably with experience in nonmetallic flotation. Location, Florida. Y6465.

Director of Engineering for an engineering institute, 30 to 40, with Doctorate in mining, chemical, electrical or mechanical engineering. Must have good experience in research, preferably industrial, and a good teaching record. Salary, $6000 to $7000 a year. Location, South. Y7166.

Engineers (a) Mine Foreman with at least three years underground experience for lead-zinc operation. Salary, $4800 a year plus housing and family accommodations. (b) Shift Boss preferably engineering graduate, with underground experience. Salary, $4200 a year plus housing and family accommodations. Must speak Spanish. Location, Peru. Y6679.

Mining Engineer, graduate, with experience in underground operations for a fluorspar mine. Salary open depending on experience. Location, western United States. Y7724.

Mining Engineer with broad experience, young, single status, and must be in excellent physical condition, as nature of work will be under very primitive conditions. Salary about 25 pct more than applicants last salary rate plus travel expenses and living allowance. Duration until July 1953. Location, Burma. Y-7498.

Mill Superintendent to take charge of a 100-ton flotation gold mill. Must speak Spanish. Elevation, 9000 ft. Salary open. Location, Argentina. Y-7216.

Columbia University
SCHOOL OF MINES
Henry Krumb Scholarships
William Campbell Fellowships

Applications are invited for the Krumb Scholarships and Campbell Fellowships for the academic year 1953-1954. These are available to candidates for degrees in the fields of Mining, Mineral, and Metallurgical Engineering.
The Krumb Scholarships, with yearly stipends of $1000 and transportation to New York City, are in support of studies toward the B.S., M.S., or professional engineering degrees. Preference is given to students in mining engineering in the ratio of three in this field to two in metallurgical and mineral engineering.
The Campbell Fellowships, established to encourage research in the general field of metals, are available to candidates for graduate and professional degrees including the Ph.D. Five grants are usually awarded each year, with stipends of $1800 for single students and $2100 for married students.
Application blanks and further information may be obtained on request to the Office of University Admissions, Columbia University, New York 27, N.Y. Complete applications should be returned before February 20, 1953.

Chief Engineer or Assistant. 42 years of age, 20 years' experience in all phases of coal mining. Will consider position in similar and/or related industry.
Box H-20 MINING ENGINEERING

"GIGANTIC WAR SURPLUS EQUIPMENT SALE"
Savings to 70%. Engines, A-C generators, welders, winches, small air compressors, water pumps, electric and gas chain saws, binoculars, many other items. Write for wholesale industrial catalog.
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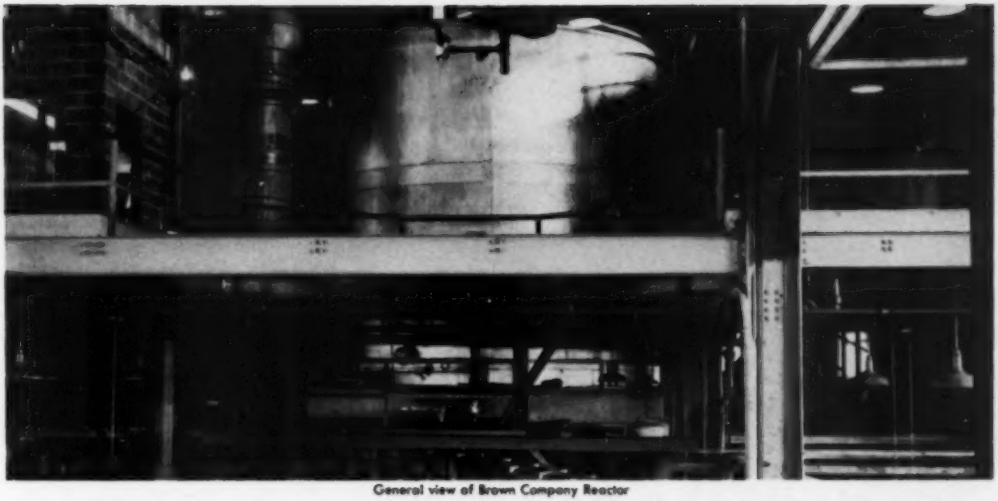
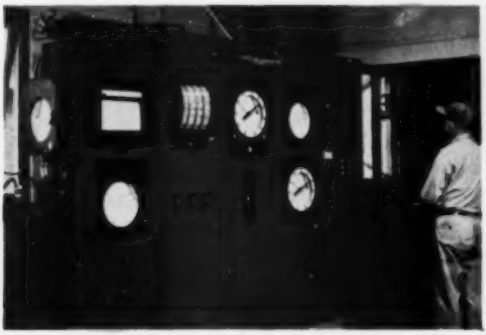
ROUGH GEMSTONE WANTED: Good quality tourmaline; yellow, purple, smoky, other colors in quartz; beryl; olivine; garnet; topaz—other materials excepting agate.
ASSOCIATED MINERS
6972 Sheridan Road, Chicago 26, U.S.A.



General view of Brown Company Reactor

First Dorrco FluoSolids* Reactor For Producing SO_2 Goes "On Stream" at Brown Company

BERLIN, N. H. In anticipation of a continuing sulfur shortage, The Brown Company at Berlin, N. H. recently installed a Dorrco FluoSolids System to produce SO_2 gas from pyrrhotite. The System at the present time is supplying the entire sulfur requirements of this sulfite pulp mill . . . 11,000 tons of sulfur equivalent per year. A producer of quality pulp for high-grade photographic papers, Brown requires clean, high-strength SO_2 gas to make their sodium sulfite cooking liquor.



Centralized control panel

RAW MATERIAL FORMERLY A WASTE PRODUCT

The pyrrhotite, recovered from tailings of a nearby copper mine, is received in a moist condition, repulped with water to 70-75% solids, and pumped into the Reactor. By operating close to theoretical oxygen requirements, the strongest possible SO_2 gas is produced, with a black magnetite calcine which is quenched and handled in slurry form. Roast is accomplished in a 16' inside diameter Reactor with the temperature automatically held at 1650°F.

Clamshell discharging pyrrhotite into blunger

HIGH STRENGTH SO_2 PRODUCED

Gas strength at the top of the Reactor averages 13% SO_2. The gas then passes through a two-stage cyclone system followed by a cooling-scrubbing tower before going to the acid towers. Average chemical analysis of feed and calcine follows:

	Total Sulfur	Sulfide Sulfur	Total Fe
Pyrrhotite Feed	35.7	35.6	49.1
Combined Calcine	0.51	0.49	60.9

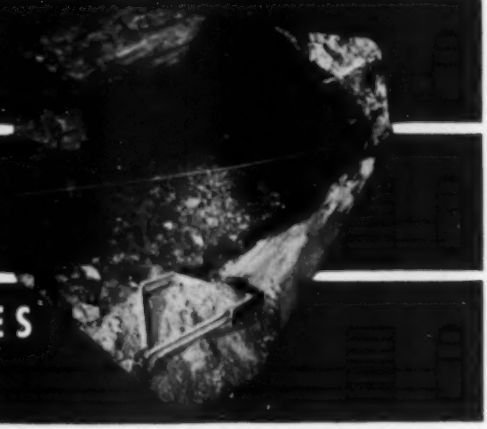
For detailed information about Fluo-Solids — a distinct departure from conventional roasters — write for Bulletin No. 7500. The Dorr Company, Stamford, Conn. or in Canada, The Dorr Company, 80 Richmond Street West, Toronto 1.

* Trademark Reg. U. S. Pat. Off.

SAVE MONEY

ON EVERY TRIP

YOUR SCRAPER MAKES



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AIR SLUSHER

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Steady power—for a full pay load every trip.
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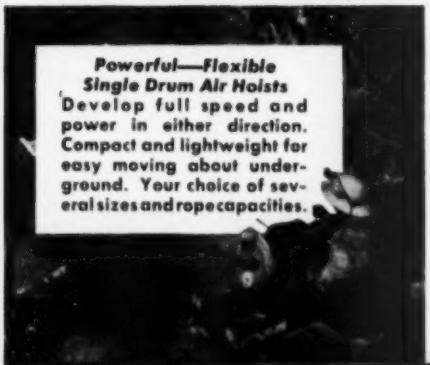
Single throttle control—readily mastered by your new miners.
Three sizes—to fit your scraper capacity efficiently.

Write today for Bulletin AS-3 on Gardner-Denver Airslushers and Air Hoists.



Powerful—Flexible Single Drum Air Hoists
Develop full speed and power in either direction. Compact and lightweight for easy moving about underground. Your choice of several sizes and rope capacities.

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Steel

BULWARK OF FREEDOM AND...

The changing pattern of progress, West of the Mississippi and East of the Rockies, is readily traced through the pages of Sheffield history.

First products of record were produced in 1888 to meet the needs of the great railroad building expansion. Following the rail lines came the fabulous development of agriculture, construction, mining, highways, manufacturing, oil and ship building.

Each of these facets of industry presented needs for steel in different shapes and forms. In meeting such kaleidoscopic requirements, the Sheffield organization acquired a vast accumulation of skills, techniques and facilities and, today, produces a wider diversity of steel products than any other similar steel making set-up in the country.

Some phases of Sheffield Steel production have expanded 3½ times in the last 10 years while Sheffield's overall production has more than doubled.

Fast and Powerful New Plate Mill—Fabricators of such steel products as pressure vessels, processing and storage tanks for liquids and gases, transportation line pipe, etc., require plate steel in far greater quantities than the slow cumbersome plate mills of yesteryear could produce. A new, powerful, 4-high plate mill at Sheffield's Houston works now rolls white hot slabs into long lengths of steel plate in a matter of minutes for shearing into sections to meet fabricators' needs.

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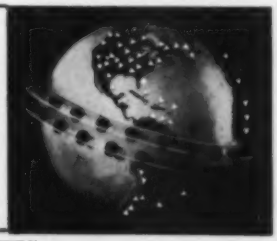
Grinding Balls
USED and PROVED ALL AROUND THE WORLD



Other SHEFFIELD STEEL Products

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Grinding Media, Forgings, Track Spikes, Bolt and Nut Products

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IF YOUR MINE IS DRY

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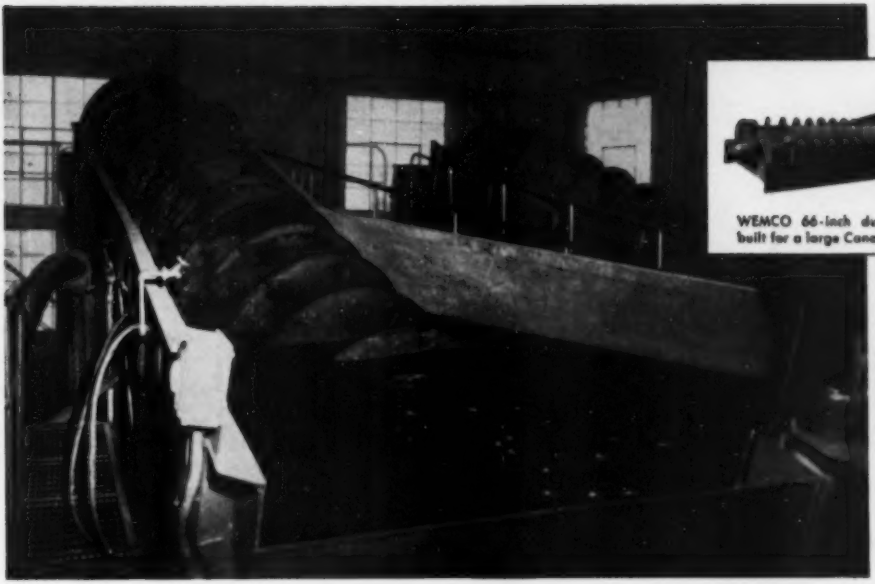
All Eimco loaders are made of ALLOY STEEL castings. The heavy-duty decks can be sealed against water or dust getting in and oil coming out. You don't even have to check the oil, in full 3 shift operation, any oftener than once a month.

In addition to this they are the fastest in loading, most dependable in operation and cheapest to operate both in maintenance and air consumption.

Check the costs of operating mechanical loaders at other mines before you decide — there's an Eimco for your size of drift and car.



WEMCO 66-inch duplex classifier built for a large Canadian operator.

Two 60-inch, series 150 WEMCO S-H Classifiers in the regrind circuit of a large lead-zinc concentrator.

for better Classifier performance

in terms of

- *Less Oversize in Overflow*
- *Less Undersize in Sands*
- *Lower Operating Costs*
- *Longer Continuous Operation*
- *Greater Capacities*

—use ***WEMCO*** *S-H Classifiers*

These Exclusive Mechanical Features *of WEMCO S-H Classifiers are among the important reasons why* ***only*** *the WEMCO unit gives you these better performance factors — with the result that more and more operators throughout the world are choosing WEMCO for their classification jobs.*

1. **40% greater shaft strength,** for protection against shutdowns, is provided by WEMCO's large diameter tubular shafting.
2. **Longer-life wearing shoes** of tough, thick alloy and large cross-section, short cast steel flight arms. Shoe replacement costs have been reported as low as 4/100 of 1 cent per ton.
3. **Foolproof, efficient bearing units** are of the anti-friction type. Lower unit is grit-proof; upper unit is gudgeon type mounted on hub of bevel gear, designed for extra long life.
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5. **Smooth, efficient drive** uses standard speed motors and a new type cone gear reducer for maximum efficiency of power transmission.

Consult with WEMCO today on your classification problems. Competent engineering of your inquiry will be handled promptly and places you under no obligation.

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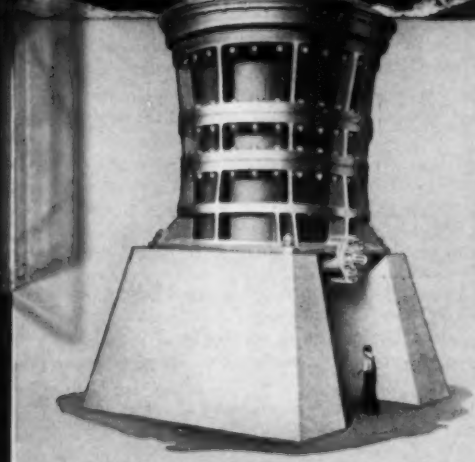


WORLD'S LARGEST CRUSHER is this 3500-tph, 60-109 *Superior* gyratory being built for Mesabi's first commercial taconite plant. Ultimate plant output will be 10 million tons.

FINES ARE SCALPED from r.o.m. ore on sturdy 1000-tph vibrating screens like this one in Minnesota. A-C builds six separate types of vibrating screens for beneficiation.

EQUIPMENT FOR IRON ORE BENEFICIATION: Gyratory and jaw crushers . . . Vibrating screens . . . All types of washers . . . Concentrating jigs . . . Rod, ball, and *Ballpeb* mills . . . Centrifugal pumps . . . Rotary kilns and coolers . . . *Texrope* V-belt drives . . . All types and sizes of electric motors and generators . . . Starters . . . Transformers and voltage regulators . . . Complete power generation, distribution, control equipment.

the Bottom!

As Mesabi Ores Lean Out, Allis-Chalmers Aids in Beneficiation

DIRECT SHIPPING ORES are fading fast up on the once fabulous Mesabi range. But despite foreign ore discoveries, by 1955 a quarter of blast furnace feed must be beneficiated domestic ore.

Allis-Chalmers — manufacturer of equipment for the steel industry from the mine to the mill — is helping lead the way to more economical beneficiation of lean ores.

Does low ore content require you to triple material handled? Allis-Chalmers will build you bigger, more efficient equipment! At far left below is the world's largest crusher, being built by A-C for a Mesabi taconite plant.

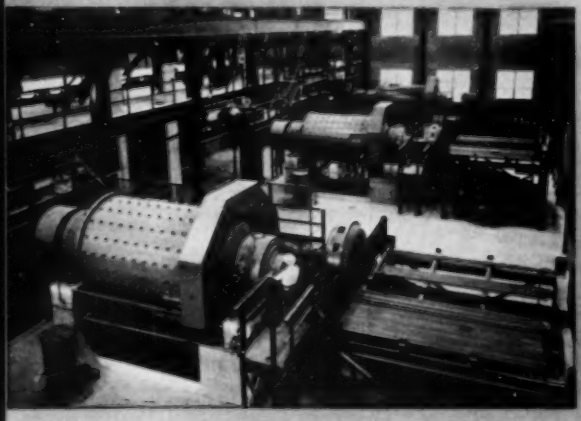
Has proper yet economical pelletizing got you stumped? Allis-Chalmers has just put into operation a pilot Pelletizing and Heat Hardening plant built exclusively for researching new pelletizing methods.

Do you have a beneficiation "shortcut" in mind? Don't let lack of equipment postpone your testing of new processes. The Allis-Chalmers Processing Research Laboratory stands ready to accurately and exhaustively test any processing method on a confidential basis.

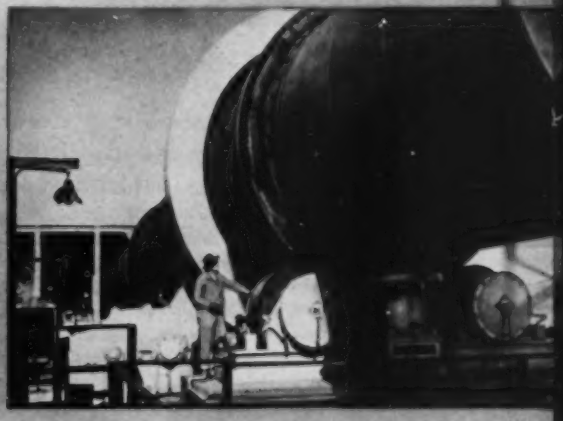
For over half a century Allis-Chalmers equipment has been processing ores for America's steel industry. Some of the cost-cutting, tonnage-increasing ore beneficiation equipment in the broad A-C line is seen on these pages.

Allis-Chalmers offers you *one-manufacturer responsibility* for process, drive and control equipment all along your flow sheet. For literature or detailed information on products and services, call your nearest A-C office or write to Allis-Chalmers, Milwaukee 1, Wisconsin. A-3808

Superior, Ballpeb and Texrope are Allis-Chalmers trademarks.



ORE IS GROUND in three rod and ball mills ahead of magnetic separation at this beneficiation plant. A-C builds *all types* of grinding mills for ore beneficiation processes.



IRON ORE IS AGGLOMERATED to make good blast furnace and open hearth feed in Allis-Chalmers rotary kilns in connection with iron mines and beneficiation plants.

ALLIS-CHALMERS

Power, Electrical, Processing Equipment for Iron and Steel

MICROMETER DEPTH INDICATORS in the Console...

(Nordberg Patent)

This Nordberg tandem, double drum underground hoist is installed in an Anaconda mine. It was designed for a depth of 1600 ft., using 8 ton skips and hoisting 6 tons of copper ore at a rope speed of 800 ft. per min.

... another reason why the Mining Industry looks to NORDBERG as the leader in Mine Hoist design and development

● SINCE 1895, Nordberg has enjoyed world-wide leadership in the hoist field, specializing in building large or unusual hoists ... and has been the source of the most progressive hoist engineering advances in the mining industry.

Micrometer Depth Indicators in the Console, for example, is just one of the advanced engineering features that has made Nordberg the leader in the design and development of modern mine hoists. This, and many other important engineering refinements, assures the very best in workmanship and materials to give you the ease of operation, safety and dependability which has made Nordberg Mine Hoists the standard for comparison.

No matter what your hoisting requirements may be, you will find that Nordberg hoist engineers are fully qualified to solve your specific problems. Write for further details, or send for Bulletin 190.

M552

NORDBERG MFG. CO., *Milwaukee, Wis.*

"SYMONS" ... a Nordberg trademark known throughout the world

NORDBERG

MACHINERY FOR PROCESSING ORES and INDUSTRIAL MINERALS

NEW YORK • SAN FRANCISCO • DULUTH • WASHINGTON • TORONTO
MEXICO D. F. • LONDON • PARIS • JOHANNESBURG

"SYMONS" Primary Crushers

Grinding Mills

Mine Hoists

"SYMONS" Vibrating Bar Grizzlies and Screens

Diesel Engines

MEET THE AUTHORS

GRANVILLE S. BORDEN

Granville S. Borden (*Mining Taxation*, P. 1049) has written two other published articles on the subject, *Taxation and Mineral Resources*, and *An Income Tax Program for Mineral Producers*. Both articles appeared in *Mining Engineering*. He is presently employed by Standard Oil Co. of California as a special tax counsel. He holds an LL. B and an LL. M from George Washington University and a Bachelor of Science in Mining from the University of California. He has been chairman of the Taxation Committee of the Minerals Economics div. of the AIME. Borden fishes, plays golf, prospects, and performs mineral analysis as hobbies.

CHARLES C. TOWLE

Charles C. Towle (co-author, *Uranium Deposits of the Grants District, New Mexico*, P. 1037) graduated from the Colorado School of Mines with a degree in Geological Engineering. He was a member of Sigma Nu; Theta Tau, an honorary professional society; Blue Key, honorary social society; the glee club and quartet, and a member of the student council. He has been a mucker, driller, flotation operator and mill shift boss for the Clear Creek-Gilpin Co., Idaho Springs, Colo.; Geologist, chief engineer, and chief geologist for Patino Mines & Enterprises Consolidated, Inc., Bolivia; appraisal engineer and part-time geology student, Denver; office geologist for Geology Dept., special Engineering div., Sea Level Canal Studies, Panama Canal; and geologist, div. of Raw Materials, USAEC, Grand Junction, Colo. He is currently chief of the Denver Exploration Branch, div. of Raw Materials, USAEC, Denver. Towle was at one time a tournament golfer, playing three and four times a week. While working in South America he played in numerous tournaments. Today, although playing only occasionally, he shoots in the low eighties.

Adrian E. Ross (*Cementing in Deep Diamond Drill Holes*, P. 1061) was born in Clintonville, N. Y. He attended Massachusetts Institute of Technology, receiving B.S. and M.S. degrees in electrical engineering. Following college he did test work with the U. S. Navy, and development work for the Electrolux Corp. Entering the Army in World War II, he commanded signal organizations and served as staff communications officer in with the Air Force, holding the rank of Lt. Col. when he left the service. Since the war he has been with Sprague and Henwood, Inc. where he is assistant to the president and chief engineer.

LEADING COPPER COMPANY

- REDUCES COSTS
- IMPROVES OPERATING CONDITIONS

WITH *MARCY* MILLS

11 - 9½'x12' Marcy Open End Rod Mills Installed as a Result of Comparative Tests

Primary grinding in this large copper mill was being done by a series of 2-stage roll-screen circuits. A 9½'x12' Marcy Open End Rod Mill was installed in parallel with them. As a result of this test 11 - 9½'x12' Marcy Open End Rod Mills have replaced all roll-screen circuits.

GRINDING DATA:

Feed: 1½" hard ore.
Grind: 4% + 10 mesh; open circuit.
Tonnage: Up to 2700 tons per day, each mill

RESULTS:

- Prepared better and more economical feed for regrind mills with greater amount of finished product.
- Lowered maintenance and operating costs by replacing complex circuits of rolls, screens, elevators and launders.
- Cleaner, healthier, safer working conditions.

Eliminate overgrinding with Marcy's principle of rapid circulation of mill content

The Mine & Smelter Supply Co.

Thomas E. Ban (co-author, *Microstructure in Iron Ore Pellets*, P. 1053) graduated from the University of Minnesota in Chemical Engineering and also completed his masters degree there under the Taconite Fellowship. He was born in Hibbing, Minnesota, where he attended high school and junior college. Now employed as a research metallurgist by the Cleveland-Cliffs Iron Co., he resides in Ishpeming, Michigan. This is his first AIME publication.

S. R. B. Cooke (*Microstructure in Iron Ore Pellets*, P. 1053), is professor of wet and mineral dressing, Montana School of Mines, but still finds time to enjoy his hobbies, astronomy, mountain climbing, and photography. He has a varied background of mining and milling experience gained in New Zealand, Australia, and Canada. He has been a research metallurgist, State Experimental Station, Rolla, Mo., assistant professor of metallurgy and ore dressing, Missouri School of Mines, and research professor of mineral dressing, Montana School of Mines. Cooke studied at the University of Otago, in New Zealand, University of New Zealand, Missouri School of Mines, Carnegie Institute of Technology, and University of Missouri. He earned a Ph.D. in 1933. A member of the AIME, he belongs to the American Chemical Society, and other scientific groups. He has authored two other AIME papers.

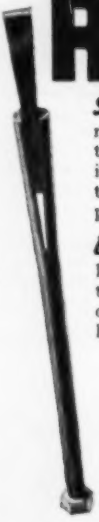
S. R. B. COOKE

IRVING RAPAPORT

Irving Rapaport (co-author, *Uranium Deposits of the Grants District, New Mexico*, P. 1037) hunts and collects folk music in his spare time, of which he intimates he has very little. He is a member of the Lions Club of Grants, N. Mex., and is a member of the AIME. At the University of Minnesota, where he received a Bachelor of Science in Geological Engineering he played freshman basketball and was secretary-treasurer of the School of Mines Society. Rapaport spent 2½ years in Europe with the paratroops. Before graduating, he worked for the Buckeye mine, Mesabi Range, in Minnesota; Climax, Colo., and the Sunshine mine, Idaho. He worked for the Atomic Energy Commission for three years as a geologist. Later, he was employed as a consultant for the Chilean Nitrate Corp., and is presently employed by Running Hanosh Mines, Inc. He is now a resident of Grants, N. Mex.

CF&I STEEL PRODUCTS FOR THE MINING INDUSTRY

ROCK BOLTS



SAFETY... Support is furnished within a few inches of the face or heading . . . blasting has little or no effect on the bolts. Affords better support than timbering.

ECONOMY... Bolts store in less space . . . handling and transportation costs are reduced. Greater permanence—less replacement.

BETTER HOUSEKEEPING ...Rock bolts provide more clearance overhead and on the sides. Ventilation is improved by the elimination of caps and posts.

. . .

Investigate the use of CF&I Rock Bolts for your own mining operation.

CF&I Products for the Mining Industry

Col-Wic Wire Cloth Screens • Mine Rails and Accessories • Rock Bolts
Wickwire Rope • Grinding Balls • Grinding Rods

THE CALIFORNIA WIRE CLOTH CORPORATION, OAKLAND
THE COLORADO FUEL AND IRON CORPORATION, DENVER and NEW YORK

CF&I ROCK BOLTS

1119

GM DIESEL
CASE HISTORY No. 526-25

USER: Aurora Limestone Products Co., Aurora, Iowa, producers of road stone and agricultural lime.

INSTALLATION: GM 3-71 Diesel powers a ¾ yd. Unit shovel, replacing a ¾ yd. shovel powered by a gasoline engine.

PERFORMANCE: GM Diesel powered shovel works faster, feeding crusher about 17% more per day. Fuel consumption averages 4 gallons per hour, saving 33% on operating costs.

THIS DIESEL LOADS FASTER

GM
GENERAL MOTORS
DIESEL POWER

Manufacturers News

New Products • FILL OUT THE COUPON FOR MORE INFORMATION • Equipment

Roof Bolts

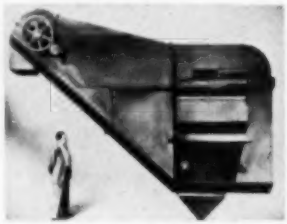
An expansion shell and plug *Westinghouse* roof bolting assembly made for use with threaded steel rods and large flat washers or steel channel employs a plug larger in diameter than those for trolley wire service. The notched plug helps prevent shell and plug from turning. The unit is shipped complete. **Circle No. 1**

Fan

The *Jeffrey Mfg. Co.* Aerodyne Midget fan is a self contained blower or exhauster for auxiliary ventilation through tubing. Operating rating with 5 hp motor is 6.5 in. maximum static pressure, and 6200 cfm free delivery volume. Weight is 340 to 475 lb and it may be furnished in open or permissible form. **Circle No. 2**

Heavy-Density

Initial installation in the Mesabi Range country of the Wilmot-Daniels heavy-density concentrating system was a 175 ton per hour plant at the



Skubic Bros. Virginia, Minn., operation. This *Wilmot Engineering Co.* unit has a separatory vessel providing unidirectional medium flow and as the only moving part uses a standard drag-line conveyor for lifting sink out of the bath. **Circle No. 3**

Portable pH Meter

A low priced battery-operated portable pH meter of utmost simplicity announced by the *Photovolt Corp.* uses only 3 ordinary radio batteries which give over 2000 hours of service. Accuracy of 0.03 pH is claimed by the manufacturer for the model 125, and AC power line supply is available. **Circle No. 4**

Pneumatic Tools

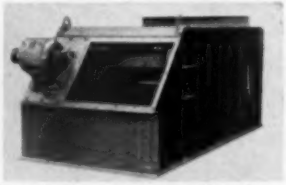
Eighty-five different items from a 1 lb 1 oz grinder to a 2 lb 11 oz angle screwdriver comprise the lightweight Thor drills, screwdrivers, nut setters and grinders made by *Independent Pneumatic Tool Co.* Parts standardization offers ready assembly and low-cost maintenance. The rotary air motor gives maximum output at standard pressures. **Circle No. 5**

Vibration

The Fowler Vibra-Tak is a pocket-size instrument for measuring speed and amplitude of vibration on machinery or on bins where vibration is intentionally produced to aid material movement. It is distributed by *Martin Engineering Co.* and has a 2000 to 15,000 cpm range. **Circle No. 6**

Grizzly

The new Symons type K-RG vibrating rod grizzly manufactured by *Nordberg Mfg. Co.* is designed to scalp in excess of 400 tons per hour.



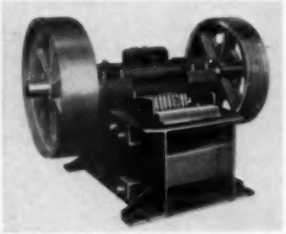
The grizzly was engineered for heavy duty service with wet, sticky or gummy rock and ore. Applicable for separations from ½ in. to about 4 in. it is suited for service following most primary crushers. **Circle No. 7**

Fire Extinguisher

One and two quart vaporizing liquid extinguishers from the *Fyr-Fyter Co.* are built for one-hand operation. Elimination of pumping and other manual operation enables the operator to aim a direct, on target stream. **Circle No. 8**

Crusher

A 10x16 in. plain bearing jaw crusher has been designed by *Pioneer Engineering Works.* A single wall pressed steel base, said to be rigid,



yet very lightweight, carries a full round bronze sleeve pitman bearing and bronze and babbitt side bearings. Capacity ranges from 4 to 40 tons per hour depending on reduction. Total weight is about 3350 lb. **Circle No. 9**

Tarpaulins

Tarpaulins made of DuPont 10 gauge Vinylite are half the weight of duck covers and are stated by *American Agency Distributors* to cost half as much. Sizes up to 22x22 ft are available. **Circle No. 10**

Conveyor Belt

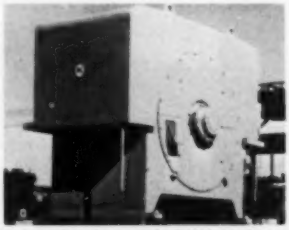
A conveyor belt claimed to be virtually rip-proof has been placed on the industrial market by the *New York Belting & Packing Co.* Designed particularly for use in mining work the outstanding feature of the belt is a special carcass which has multiple strands of high tensile steel wires at three-foot intervals. **Circle No. 11**

Pipewrap

The anti-corrosion qualities of poly-vinyl chloride and the ease of application of pressure-sensitive tape have been combined in two new tapes manufactured exclusively for pipewrapping by *Minnesota Mining & Mfg. Co.* **Circle No. 12**

Motor

An outdoor weather-protected motor carrying protection much further than splashproof design is available from *Allis-Chalmers Mfg. Co.* for semioutdoor and outdoor stations.



Designed to keep wind-driven heavy moisture out of the windings, the intake air velocity has been reduced to less than 600 fpm. **Circle No. 13**

Raintight Switch

Featuring front operation for close ganging, a new switch designed for outdoor installation by *Federal Electric Products Co.* is stated to be the only raintight switch on the market for front operation. **Circle No. 14**

Soldering Iron

Requiring no electric current or external heat of any kind, the Quikshot soldering iron utilized a chemical cartridge that heats the iron to working temperature in 10 seconds and maintains 800°F temperature for seven minutes. It is suitable for work where line power is unavailable. *Kenmode Mfg. Co.* Circle No. 15

Pumps

Frame type centrifugal pumps designed to handle most ordinary pumping jobs requiring capacities up to 500 gpm and heads as high as 135 ft are available from *Allis-Chalmers Mfg. Co.* The pumps are designed for V-belt drive, and for operation in a variety of positions. **Circle No. 16**

Free Literature

FROM AIRPLANES TO TRUCKS.... Modern electronic type scales produced by Cox and Stevens Aircraft Co. are the outgrowth of 10 years' experience electronically weighing aircraft. Capacity is up to 50 tons for trucks and up to 400 tons for railroad cars. Key unit is a group of 4 to 6 electronic cells, but 6 in. high, which work without levers and require a very shallow pit for installation.

(17) WORTHINGTON: Cartoonist-author Don Herold recently took a look at the *Worthington Corp.*, putting the results into a compact booklet called "A lot of things beside pumps." The story is aimed to better acquaint you with Worthington and how they came to produce a diversified industrial equipment.

(18) SPEED REDUCERS: Now available is *Cleveland Worm & Gear Co.'s* new general catalog. In its 180 pages are the illustrations and description of the complete Cleveland line, together with full engineering data, dimensions, weights and horsepower capacities of the various drives.

(19) METAL PRODUCTS: *Armco Drainage and Metal Products, Inc.* has just published a condensed 6-page catalog of its products designed for mining use. It describes and illustrates such products as Flex-Beam guardrails, corrugated metal pipe, liner plates, retaining walls, Steelox buildings, and welded pipe.

(20) RECORDS: Pre-scheduled, controlled, preventive maintenance reduces costly downtime to a minimum and eliminates the need for large emergency crews. A new folder describes how such a program can be accomplished by using *Remington Rand Inc.*, Kardex visible records. Another Remington Rand Inc. bulletin is the certified case history of how Kardex visible purchasing records help maintain a smooth running purchasing department.

(21) FLOTATION: Two bulletins now available from *Denver Equipment Co.* are "The Unseen Factors in Flotation," a 4-page article on the factors in flotation which affect economic recovery; and "Methods and Machines Used in Ore Testing," by Clarence Thom, director of Denver's ore testing activities.

(22) SUBSTATIONS: Ignitron rectifier substations are the subject of a *Westinghouse Electric Corp.* bulletin. It discusses efficiency, performance and convenience of ignitrons vs. M-G sets, and describes the operation of portable and stationary ignitron substations.

(23) DIESEL APPLICATIONS: The fall issue of Power Parade, from *Detroit Diesel Engine Div., General Motors Corp.* deals with interesting engineering applications in industrial, marine and petroleum fields;

using pictures and text to tell the story of successful diesel installations. The photo shows a Plymouth locomotive, newly repowered with a 3-cylinder GM diesel, hauling twice its former load at 27 pct of former fuel cost.

(24) HARDFACING: *The Stoody Co.* manufacturer of hard-facing alloys, have revised their Stoody Hard-Facing Guidebook, available to all those interested in the rebuilding and hard-facing of heavy equipment.

(25) BULLDOZERS: Recent bulldozer operations in many areas are presented in "Pay Dirt Push" a booklet published by *Caterpillar Tractor Co.* The booklet tells how the selection of bulldozer design and adjustments can contribute to job

efficiency. Examples are given of the application of cable and hydraulic controlled blades—straight, angling-type and U-shaped. The photo shows a Cat D6 feeding iron pyrite ore to conveyor at Mountain Copper Co. mine Matheson, Calif.

(26) CHANGEROOMS: "The Design and Layout of Industrial Changerooms" is a 16 page illustrated booklet written for the design engineer, safety engineer, and others with changeroom problems, by the *Moore Co.* Architectural standards, dimensioning, equipment, and safety are emphasized throughout, providing easy reference for the planning engineer.

(27) PUMPS: *Ingersoll-Rand Co.* has released a new bulletin covering general purpose centrifugal pumps of the cradle-mounted type. It covers 5 basic cradle groups and 17 corresponding pump types, their capacities, horsepower ratings and uses. There are also two pages describing special purpose units and designs.

Mining Engineering **November**
29 West 39th St.
New York 18, N. Y.

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items indicated.

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
21	22	23	24	25	26	27	28	29	30

Students are requested to write direct to the manufacturer.

Name_____ Title_____

Company_____

Street_____

City and Zone_____ State_____

DR34 feed and H23DR drill mounted on truck chassis. Note that Le Roi-AIRMASTER compressor is also mounted on chassis, to supply plenty of low-cost air.

Uranium operation gets remarkable performance from LE ROI-CLEVELAND AIR FEED and H23DR DRILL

HOLES as deep as 120 feet drilled with sufficient air for hole cleaning! 18 feet of hole in 12 to 14 minutes, with 90 to 95 psi air pressure. Mister, that's real drilling! And it's typical of the money-making performance you get with Le Roi-CLEVELAND rock drills.

The DR34 air-feed shell provides a 21' feed with 20' steel change. The drill is a Le Roi-CLEVELAND 80-lb. H23DR with 5/16" blowing, strong rotation, positive valve action, plenty of wallop, and dependable durability. Chuck size is standard — 1" hex, 4-1/4" shank. Octagon or round-lug shanks can be used, also.

Though 120' holes have been sunk in uranium-prospect drilling operations — using jointed steels and standard forged steel-type detachable bits with stellited corners — average drilling depth with the DR34 for blast holes is 40' to 50', depending on the contour of the terrain.

On deep-hole work, enjoy more footage per shift — do your drilling better, faster, for less — with Le Roi-CLEVELAND machines.

Write today for complete information.

LE ROI COMPANY

CLEVELAND ROCK DRILL DIVISION
12500 Berea Road, Cleveland 11, Ohio
Plants: Milwaukee, Cleveland and Greenwich, Ohio

RD-48

Two men on a tredmill

worked hard to grind a small quantity of ore with this 16th century rig. A historical curiosity today, it was once considered a highly efficient machine.

The Traylor TC Gyratory, with its curved concaves and ball head, incorporates proven principles of advanced crusher design. Bulletin 126 illustrates its superiority in detail.

A comparison of the past with the present illustrates the great progress made in finding new ways to do a job better . . . make work easier. For 50 years, Traylor has taken a leading part in developing more efficient machinery for crushing and grinding ore. By working closely with the mining industry, we have "Traylored" our equipment to keep ahead of its most exacting demands. The key to designing outstanding machinery is experience. Traylor has experience . . . half a century of it.

OUR GOLDEN ANNIVERSARY YEAR

1902 1952

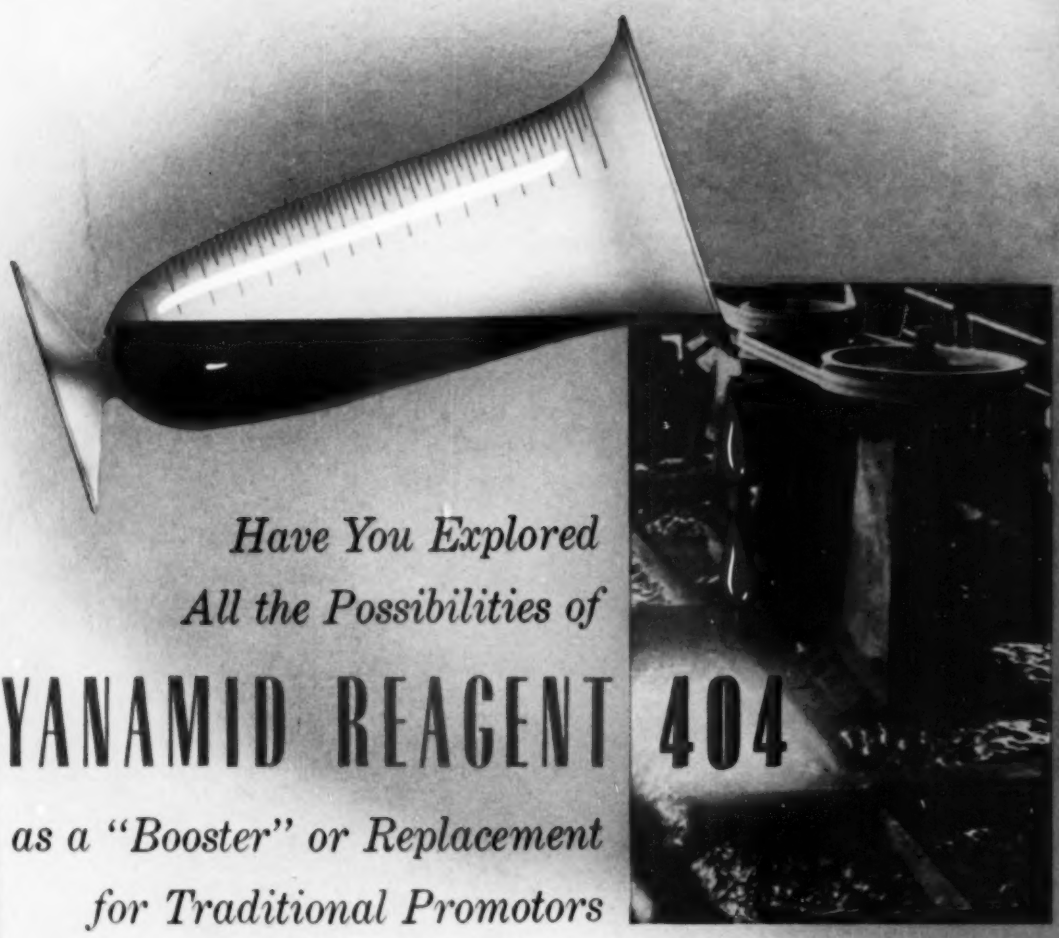
TRAYLOR ENGINEERING & MANUFACTURING CO.

1424 MILL ST., ALLENTOWN, PA.

SALES OFFICES: New York, Chicago, San Francisco

Canadian Mfr.: Canadian Vickers, Ltd., Montreal, P. Q.

a **Traylor** *leads to greater profits*



Have You Explored
All the Possibilities of

CYANAMID REAGENT 404

as a "Booster" or Replacement
for Traditional Promotors

Challenging time-honored ideas on reagent combinations, more and more metallurgists are sharpening selectivity, increasing recovery and reducing reagent costs by introducing Cyanamid Reagent 404 in their flotation circuits.

Fastest growing among the newer reagents in Cyanamid's complete line of metallurgical chemicals, Reagent 404 is already being used alone and in combination to beneficiate auriferous pyrite and pyrrhotite, tarnished and oxide lead, lead-zinc, copper-zinc, copper-lead, free gold, cobalt, antimony as well as other complex precious and base-metal ores. Each year, Reagent 404 finds new applications beyond its anticipated sphere of use.

Consider these four typical examples of the introduction of Cyanamid Reagent 404 as a "booster" or replacement for less effective promotors:

AMERICAN Cyanamid COMPANY

COPPER—PYRITE ORE, SILICIOUS GANGUE

PROBLEM: To reduce insoluble content of concentrate and increase copper recovery.

SOLUTION: Using Reagent 404 as part replacement of previous promotor produced these comparative test results in the mill.

	PROMOTOR lbs./ton	FROTHER lbs./ton	REAGENT COST	CONCENTRATE % Cu.	% Fe O	% Insol.	TAILING % Cu.
Previous Promotor X	.05	0.20	5.67¢	25.0	28.6	16.2	0.254
Present Promotor X	.025	0.045	3.29¢	24.6	31.6	12.3	0.206
Reagent 404	.032						

RESULTS: Reagent 404 lowers insoluble in concentrate and cuts reagent costs 42% (2.38¢ per ton of feed.)

PARTIALLY OXIDIZED LEAD ORE CONTAINING GALENA AND CERUSSITE

PROBLEM: To simplify the flow sheet and improve recovery of lead.

SOLUTION: Use Reagent 404 in place of usual type promotor and sulfidizing agent.

Product	ASSAY % Wt	% Pb	% H-S Pb	% Distribution of Lead
Feed	100.0	11.05	2.91	100.0
Concentrate	16.6	65.0	— —	97.6
Cleaner Tailing	4.4	2.33	— —	0.9
Rougher Tailing	79.0	0.30	0.16	1.5

RESULTS: Excellent metallurgy with simple flow sheet and reagent combination. Water reclamation and circulation entirely feasible since Reagent 404 does not disperse tailing and thus interfere with thickening as would Sodium Sulfide. Galena and cerussite are floated in a single circuit, as compared with the necessity for two separate flotation circuits when a sulfidizing agent is used to recover cerussite.

AURIFEROUS PYRITE WITH NO FREE GOLD. CLEAN SILICIOUS GANGUE

PROBLEM: To increase recovery and reduce reagent cost.

SOLUTION: Use Reagent 404 as total replacement for promotors previously used.

	PROMOTOR lbs./ton	FROTHER lbs./ton	REAGENT COSTS	ASSAYS, Au, Oz. T FEED	CONC.	TAIL
Previous Promotor A	0.10	0.07	5.23	0.129	5.16	0.006
Promotor B	0.05					
Present Reagent 404	0.05	0.05	2.60	0.175	5.07	0.005

RESULTS: Reagent 404 increases recovery of copper, lowers tailing by 3½¢.

GOLD—SILVER—LEAD—ZINC ORE

PROBLEM: To increase recoveries of gold, silver, and lead in the lead concentrate.

SOLUTION: Use of Reagent 404 as part replacement of previous promotor combination gave these comparative results.

	PROMOTOR lbs./ton	FEED, Oz./ton Au	Ag	% Pb	Pb Conc. Oz./ton Au	Ag	% Pb	Recoveries, Pb Conc. Au	Ag	Pb
Previous Promotor A	0.10	0.13	4.2	2.5	0.80	36.0	24.7	43.0	56.8	65.1
Present Promotor A	0.04									
Reagent 404	0.02	0.14	4.4	2.6	0.83	31.1	23.3	51.8	59.4	71.6

RESULTS: Use of Reagent 404 increased the recovery of Gold 8.8%, the recovery of silver 3.4% and the recovery of lead 6.5%. Simultaneously, reagents costs were reduced by 1¢ per ton.

An objective review of your reagent combinations and treatment methods in the light of recent milling developments may reveal that Cyanamid Reagent 404 (or its companion Reagents 425 and 444) can help you to achieve lower cost-plus-tails on your present ore now and make it economic to treat lower-grade mill feed in the future.

Cyanamid Field Engineers and Cyanamid Mineral Dressing Laboratory keep constantly informed on metallurgical chemical advances and flow-scheme modifications in every mining field the world over. They stand ready to give you unbiased counsel and practical help in your reagent studies... to bring you the benefits of Cyanamid's fundamental research, testing facilities and on-the-mill experience in applying more efficient reagents and processes. Your inquiry is invited.

MINERAL DRESSING DIVISION 30 ROCKEFELLER PLAZA NEW YORK 20, NEW YORK

Printed in U.S.A.

See for yourself how Ni-Hard fittings fight abrasion, and cut costs

THESE NI-HARD FITTINGS for HUDSON COAL CO., were produced by Abrasive Alloy Casting Co. Ni-Hard applications include: grinding balls, liners, slurry pump parts, flotation impellers, piping, scoop lips, pulverizer rings, classifier shoes, roll heads, etc.

Ni-Hard® . . . one of the hardest engineering materials available . . . is a moderately priced nickel-chromium white cast iron.

Wherever Ni-Hard is used, its remarkable abrasion resistance not only helps to minimize replacements and repairs, but also helps to reduce production costs.

Shown above are Ni-Hard fittings, cast by the ABRASIVE ALLOY CASTING COMPANY of Bridgeboro, New Jersey, for use in discharge lines of Chance Separators operated by THE HUDSON COAL COMPANY of Scranton, Pa.

Heretofore, Hudson Coal used extra thick gray iron fittings for this service.

But the Ni-Hard fittings . . . installed in 1950 to handle the highly abrasive sand, waste and water mixture at 3,000 GPM . . . now appear capable of lasting three to four times longer than the thicker sectioned gray iron fittings.

Use Ni-Hard on *your* tough jobs . . . for lowest ultimate cost.

Investigate Ni-Hard. Get full information . . . mail the coupon now.

At the present time, the bulk of the nickel produced is being diverted to defense, but nickel is obtainable for the production of Ni-Hard for many end uses in defense and defense supporting industries. There are authorized producers, from coast to coast, equipped to produce Ni-Hard castings in all common forms and shapes.

INCO

THE INTERNATIONAL NICKEL COMPANY, INC.
Dept. 20, 67 Wall Street, New York 5, N. Y.
Please send me free booklets entitled: "Engineering Properties and Applications of Ni-Hard" and "Buyers' Guide for Ni-Hard Castings."
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Company _____
Address _____
City _____ State _____

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N.Y.

September steel production established a new record for a 30-day month with 9,034,000 tons. It was the highest of any month since March. Output of ingots and steel for castings was 535,000 tons more than was poured in August 1952, and 373,000 tons over the total for September 1951.

Prime Minister Daniel F. Malan touched a gold button to start South Africa's first uranium plant into production on the premises of the West Rand Consolidated Gold Mine, near Krugersdorp, Transvaal, 25 miles west of Johannesburg. Uranium oxide is being extracted from mine tailing after removal of the gold. Thirteen gold mines in the region are scheduled for uranium production.

Defense Materials Procurement Agency signed an agreement with the Bagdad Copper Corp., Bagdad, Ariz., for expansion of the firm's facilities to enable delivery of at least 27 million lb of copper a year and 940,000 lb of molybdenite. It is said that the company will spend about $14 million on the expansion.

World tin buyers were awaiting formal action by the Bolivian Government for the nationalization of that country's tin mines after Government action resulted in seizure of the mines of Bolivia's three biggest companies, Aramayo, Patino, and Hochschild companies.

Jess Larson, administrator of General Services, announced that the Government will buy beryllium ore at $400 per ton on visual inspection. To be accepted, ore must contain not less than 8 pct beryllium oxide by weight and must be in the form of clear crystals, cobbed free of waste.

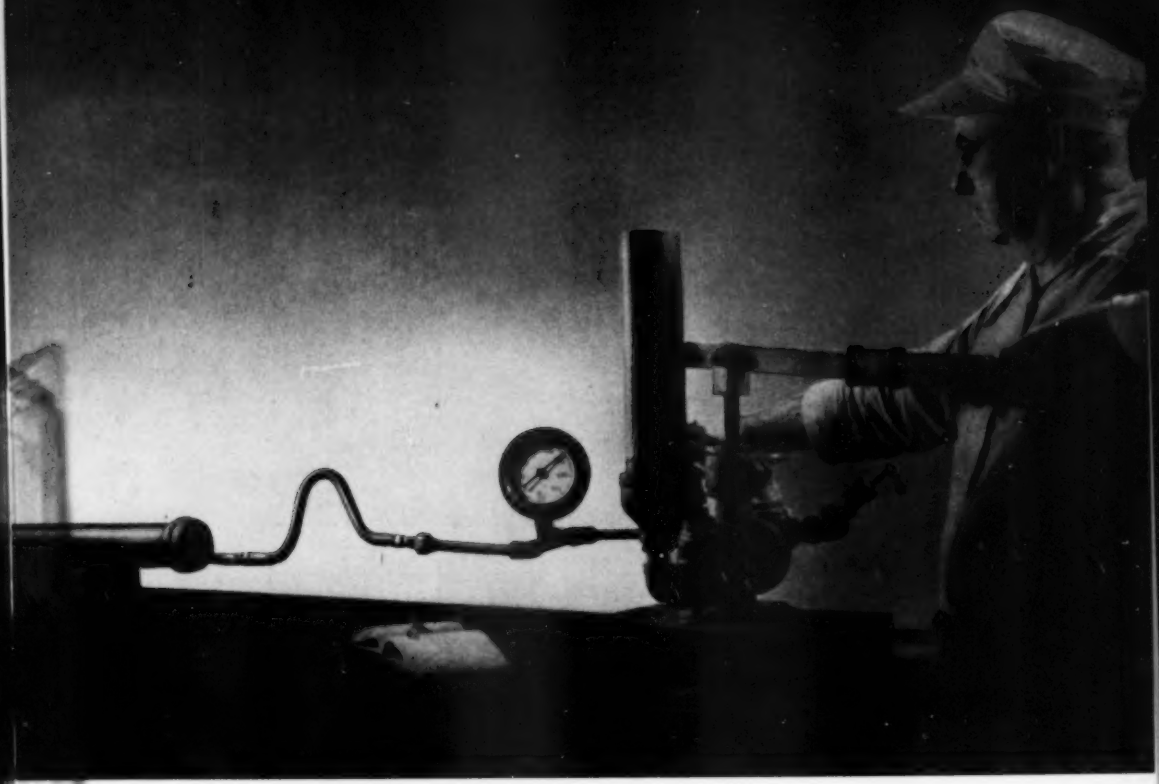
Wyoming is the first state to enter into an agreement with the Federal Government under the provisions of the Federal Coal Mine Safety Act. Under the terms of the agreement, Federal and State inspectors will make joint inspections of Wyoming mines except where participation by State agents might tend to unduly slow action in cases of extreme danger.

An all-day titanium symposium, held at Watertown Arsenal, Mass., urged increased civilian use of the new metal in order than the price be driven down and production facilities increased. The Army, at present, is using about 90 pct of total production.

Steep Rock Iron Mines, Ltd., announced work on a $2.5 million addition to the Port Arthur ore dock and a $2 million block signal system for the Port Arthur-Atikokan line of the Canadian National Railways, beginning the next few months. Construction is part of a program to double iron ore shipments from Steep Rock by 1955.

Rem-Cru Titanium, Inc., announced that it will expand titanium production, eventually doubling production capacity. New melting facilities now being installed at Midland, Pa., are scheduled for completion this year. New furnaces and changes on existing furnaces are expected to boost ingot size to 2000 to 4000 lb.

PROGRESS IN EXPLOSIVES . . .



FOR WORKING UNDER PRESSURE

Hercules' research has developed explosives that shoot successfully even under the severe water pressures encountered today in seismograph prospecting.

In the laboratory test shown here, a cartridge of Hercules Vibrogel* 3 is sealed in the steel "bomb" at the extreme left. It then undergoes prolonged exposure to pressures greater than those ordinarily found in actual field use. Because Vibrogel 3 embodies a new conception of explosives research, it detonates despite these extreme conditions.

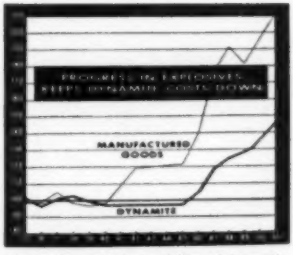


Chart shows relative stability of dynamite prices since 1935, as compared with prices of other manufactured goods. 1935-39 values=100.

HERCULES POWDER COMPANY
INCORPORATED

Explosives Department, 955 King St., Wilmington 99, Del.

U. S. Signs Pact For Bolivian Tin

After months of negotiations, the U. S. has agreed to buy between 6000 and 7000 long tons of Bolivian tin now stockpiled at Peruvian and Chilean ports, at $1.175 per lb, f.o.b. South American Ports.

Harry A. McDonald of the Reconstruction Finance Corp. signed the contract with Victor Andrade, Bolivian ambassador. The Mercantile Metals and Ore Corp., of New York City, acted as the agent of Bolivia's Banco Minero in negotiating the deal.

The contract is a short term one. Throughout the talks, the U. S. refused to make a long term agreement because of the possibility that Bolivia will soon nationalize its mines. The Bolivians desired to sell the U. S. 18,000 tons during a one-year pact.

The price agreed upon is roughly the same as with Malayan producers.

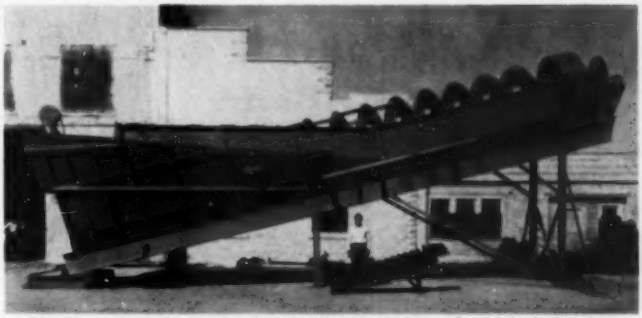
In the meantime, American Smelting & Refining Co., signed over control of its operations to the Bolivian Government. The formal act of leasing the mines to the Bolivian Government follows the announcement last July by the company that it was closing operations because of high costs. Contract terms were not disclosed.

McCarthy Signs Mining Contract with Bolivia

Glenn McCarthy, oil operator, will invest $3 million in a sulphur mining concession in southwestern Bolivia under terms of a contract signed with the Bolivian Mining Bank.

Bank Manager Manuel Barrau said that the investment may be increased if necessary. The Bolivian Government will receive 8 pct royalty on gross sulphur exports.

Work on the concession, in the Potosi Department on the Chilean border, is expected to begin in about six months.



This 84 in. spiral rake classifier built by Western Machinery Co., is the largest ever built by the firm for the Mesabi Iron Range. Its hugeness is illustrated by comparison with the almost lilliputian pilot plant classifier.

U. S. To Purchase More Canadian Nickel

A U. S. Government contract with three Canadian companies promises to add significantly to the nickel supply available in the U. S. Nickel is among the most critical metals on Defense Production Administration's list.

One agreement provides that East Rim Nickel Mines, Ltd., will mine and crush ore from its Sudbury district property at a rate of not less than 3500 short tons per month. DPA has contracted for up to 65,000 tons.

Falconbridge Nickel Mines, Ltd., will process the ore at its Falconbridge, Ontario, and Kristiansand, Norway plants. East Rim, with a total expenditure of $1,000,000 for its mines, has no processing plant, although processing units are scheduled for installation. East Rim ore is expected to yield appreciable copper.

Another agreement, also involving Falconbridge, is with Milnet Mines, Ltd., of Toronto. Milnet, under the agreement, will begin mining properties in the Sudbury district no later than Mar. 1, 1953, producing ore at the rate of 2100 short tons per week to the end of 1954. Falconbridge will process the ore. Jess Larxon, Administrator of DMPA said the agreement with Milnet and Falconbridge will result in the production of 4,104,000 lb of nickel. DMPA will lend Milnet $440,000, repayable with interest as ore is produced and sold to Falconbridge.

Copper Buy Pact Signed

The Government has signed a floor price purchase agreement with the San Manuel Copper Corp., insuring development of the company's holdings in Pinal County, Ariz. The deposit is said to be one of the largest in the nation.

The Government will pay 24¢ per lb for a total of 365,000 short tons of electrolytically refined copper. The agreement also calls for payment of 60¢ per lb for 16,060 tons of molybdenum concentrates. RFC recently approved a $94 million loan to San Manuel.

High Operating Costs Shuts Two Coal Mines

Two coal mines, one in West Virginia, and the other in Tennessee closed down almost immediately following the signing of a new contract between coal operators and John L. Lewis' United Mine Workers.

Diamond Coal Co.'s Royal Blue coal mine near Caryville, Tenn., was one of the mines closed down. The mining town for years was a model for coal mining companies seeking better living conditions for their workers. Blue Diamond officials gave no reason for the shutdown, but it was learned that the mine had been operating at a loss, and the new pact between southern operators and the union appeared to be the spark.

The Christopher Coal Co., at Morgantown, W. Va., closed its Pursglove No. 8 mine, laying off about 230 men.

To Process Low-Grade Manganese Ore-Deposits

Manganese Chemical Co. plans to build a $2 million processing plant for the recovery of manganese from low-grade deposits on the Cuyuna Range in Minnesota. The plant site will be near Riverton.

It has been reported that the Dean chemical leaching process will be used rather than gravity concentration. It is expected that the plant will be able to treat at least 200 tons of manganiferrous iron ores per day.

Defense Procurement Agency will advance up to $1.5 million for construction, with repayment starting when the plant is in operation. The Government has also agreed to buy at the floor price any manganese that cannot be sold to private industry.

Annual capacity of the plant is estimated at about 546,000 long ton units of manganese concentrates containing not less than 60 pct manganese. The contract covers a total of 2,625,000 long ton units.

Manganese Chemical has been doing experimental work on the Dean process, for which it has acquired patent rights.

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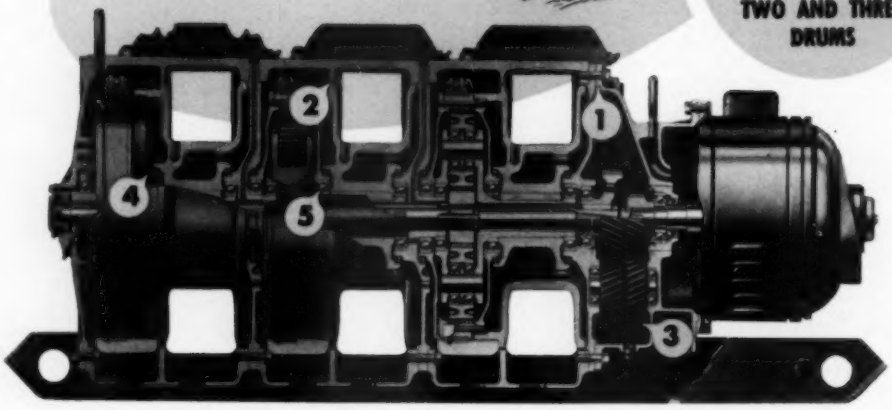
153 models

UNIT CONSTRUCTION TWO AND THREE DRUMS



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U. S. Vanadium's Uravan, Colo. Mill Doubles Output

Colorado Plateau Scene of Stepped-Up Exploration

Symbolic of the gigantic development of the Colorado Plateau, the U. S. Vanadium Corp.'s Uravan, Colo., mill recently put into operation new equipment which is expected to double its capacity. In conjunction with the expansion program, a new process used for treating high-lime ores is reported to be the most efficient for uranium-vanadium carnotite ores of the Plateau.

High-lime ores are reasonably plentiful in the region, but have never been mined because of economic factors. U. S. Vanadium believes the new process and installations, will open new vistas, and many of the difficulties attached to using high-lime ores will be overcome. The probability is that the process will bring increased mining activity in the area. It involves a roasting, leaching and precipitation cycle.

After chemical reagents, such as salt, are added, the ore is fed into roasters several stories high. The ore is then leached, with the uranium and vanadium dissolved by alkaline solutions. Each mineral is then precipitated by further processing procedures. For some ores, roasting may not be required.

U. S. Vanadium spent approximately $6 million during the past year for payrolls, supplies, and materials, exclusive of ore. It is estimated that more than $30 million a year is being spent for the mining and refining of ore throughout the Colorado Plateau, with almost 500 government employees taking part in this phase of the atomic energy program. More than 1000 people are in the mining end, and 250 are engaged in drilling operations. At least 1000 are in the milling operations.

Exploration is being carried at almost a peak rate. Private mining companies are reportedly spending more than $3 million a year for drilling alone. More than 2 million ft of drilling a year has been done, with the Government itself engaged in what has been termed a tremendous drilling project, through private companies.

The large uranium ore treatment plant of U. S. Vanadium Corp., a subsidiary of Union Carbide and Carbon Corp., at Uravan, Colo., is set below the mesa overlooking part of San Miguel Valley. Seven other mills are in operation on the Plateau, with two more under construction.

Upwards of 200 mines are being worked in the area, with mine operators battling rugged transportation problems daily. Truckers are often forced to make two trips a day over several hundred miles of rough roads that are little more than winding wagon trails. The Atomic Energy Commission is sponsoring a $4 million road construction project, with more than $2 million planned for Colorado. Private contractors are doing actual construction work.

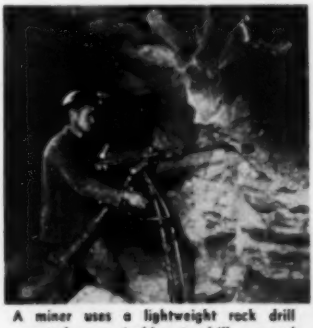
United States Vanadium has about 50 active agreements with mine contractors for the working of its properties, and also buys ore from about 90 independent contractors.

Of the eight uranium refining mills on the Plateau, two are government-owned. They are located at Durango, Colo., and Monticello, Utah. Privately-owned plants are at Grand Junction and Naturita, Colo.; and Hite and Salt Lake City, Utah. Two new mills are under construction at Grants, and Shiprock, N. Mex.

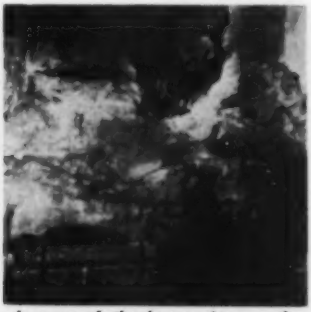
A Plateau uranium mine usually covers an area of between 20 to 40 acres. In most cases, mining consists

(Continued on page 1026)

Typical of the small scale operations in most of the mines on the Colorado Plateau, two miners hand tram in small car. Wheelbarrows are also often used.



A miner uses a lightweight rock drill mounted on a jackleg to drill a round through solid sandstone under typical conditions on the Colorado Plateau area.



In some of the larger mines on the Plateau, it is possible to use mucking machines to load ore into cars. One miner shovels ore in the stope above.

of simply following the orebody in its serpentine path through masses of waste rock. At some mines shafts may vary from 12° to the vertical. Little timber is needed because of the solid rock, but water supply poses a problem. It often has to be hauled from streams miles away, or obtained from deep wells. Many of the mines have chutes which send the ore over the cliff to a bin below, where it will eventually be picked up by a truck.

The expansion at Uravan included the erection of thickening and solution tanks of about 40,000 gal capacity; new roasters, additional filtering equipment, and enlargement of the sampling plant. The expansion comes at a time when the Government is asking greater domestic production.

Sweden Stops Iron Ore Exports to Czechoslovakia

Two iron curtain nations have had Swedish ore supplies denied them on the grounds that their indebtedness for previous shipments has reached the maximum point allowed by trade agreements.

Announcement that Sweden will no longer ship iron ore to Czechoslovakia came from a foreign office spokesman. He said the stoppage was for purely economic reasons. Czechoslovakia's payment debt reached $5,790,400, the limit allowed by treaty. The agreement called for 500,000 tons of Swedish ore, compared with 796,000 tons last year. The greatest part of the amount has been delivered, according to the spokesman.

Iron ore shipments have already been stopped to Poland, whose debt reached $4,830,840. Poland's situation has been further aggravated because of Sweden's refusal to buy Polish coal, which represents about 95 pct of Poland's exports to Sweden. Difficulties over fixing price reductions led to the move.

Speed Construction On Sherritt Gordon Refinery

Construction workers on the $17.5 million refinery of Sherritt Gordon Mines, Ltd., are working double shifts, six days a week, in order to have the plant in operation by 1954.

More than 200 men, engineers, carpenters, truckers and other workers are engaged in the project 18 miles northwest of Edmonton. The C.N.R. is scheduled to build a spur line to transport ore from Sherritt Gordon Mines at Lynn Lake to the refinery.

Yugoslavia-Germany In Trade Agreement

Iron ores and various nonferrous metals are being delivered to West Germany under a new trade agreement pact signed between Germany and Yugoslavia.

Iron ore and nonferrous metals deliveries will amount to about $13.83 million. Germany is committed to sell 26,800 tons of hot rolled steel to Yugoslavia, in addition to other finished products. Coal deliveries have been set at 290,000 tons.

Johns-Manville Joins In Columbian Project

Johns-Manville Corp., is part-owner of a newly formed company which will explore asbestos deposits in the Department of Antioquia, Columbia. Partners in the venture are Eternit Columbiana, S. A., Sociedad Columbiana de Asbestos Ltda., and the Instituto de Fomenta Industrial, a Columbian Government group.

If exploration is successful, Johns-Manville will design a mill and other facilities. Research facilities also will be provided at Manville, N. J.

No Pact Between Newmont-Japanese Firm

Newmont Mining Co. has denied that it has entered into a formal contract with Nihon Yakin Kogyo K. K. (Japan Metallurgical Works) for pilot plant production of nickel matte.

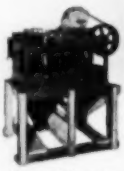
A Japanese report of an agreement whereby Newmont would advance Nihon Yakin $2 million for a pilot plant was termed false by a Newmont representative.

Denver-Dillon Vibrating Screens

Denver Ore Feeders

Jaw Crushers Denver

Denver Steel-Head Ball Mills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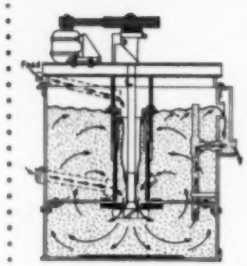
Denver SRL Pumps

Note how recirculation ports in standpipe eliminate short-circuiting. Adjustable collar on standpipe gives better recirculation control.

Shutdown is no problem. Rubber-covered wearing plate prevents sanding up.

Two of the reasons why so many mills use DENVER Super-Agitator and Conditioners are: (1) more substantial construction (2) greater flexibility in operation.



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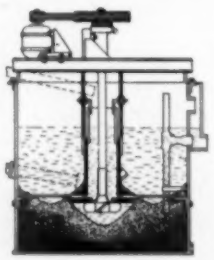

Denver Mechanical Gold Pan

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Venezuela to Reopen Expropriated Gold Mine

The National Mines of El Callao, expropriated in 1950 by the Venezuelan Government from the Canadian-owned Guayana Mines, Ltd., will be back in production by the end of the year, according to the Ministry of Mines and Hydrocarbons.

The Government paid a little over $3 million for the property after labor difficulties resulted in the closing. Venezuelan gold production reached 46,808 grams, for the first half of 1952, compared with 44,857 during the same period last year. Total output during 1951 was 88,972 grams.

Venezuelan Iron Ore Production Increases

Venezuelan iron ore production for the first half of 1952 amounted to 919,986 tons, for an average tonnage of 153,331 per month, according to the Ministry of Mines and Hydrocarbons.

Production last year for the comparative period was 534,776 tons, or an average of 89,130 tons per month. Production for 1952 came from the El Pao mines of the Iron Mines Co. of Venezuela, Bethlehem Steel subsidiary. Exports to the U. S. totaled 796,730 tons.

Corpus Christi Channel To Aid Ore Deliveries

A contract for the dredging of a deep-water channel approximately five miles long across Corpus Christi Bay in Texas, has been negotiated by Reynolds Metal Co., in connection with the La Quinta alumina plant now under construction on the bay, 14 miles from Corpus Christi.

The channel will permit company carriers to bring ore from Jamaican mines now being developed by Reynolds. The channel work will be undertaken by the Atlantic Gulf & Pacific Co., of New York.

When the alumina plant is finished, it will be one of the few installations in the world where bauxite enters on one side and emerges on the other as primary aluminum.

Small Producers Share Iron, Tungsten Expansion

Small mine companies will share in two industrial expansion goals—iron ore and tungsten ore—according to the Small Defense Plants Administration.

The small business share of iron ore expansion has been set at 24 pct of the established goal of 147 million gross tons in 1955. For this purpose small business firms have been designated as those employing 800 or less.

Small business will also get 15 pct of the 40 million lb expansion goal of contained tungsten in 1954, which includes both domestic production and imports. Here, small business has been pegged at 75 employees or less.

Small producers who wish to participate in the program must file applications for certificates of necessity at the nearest field office of the National Production Authority, U. S. Department of Commerce. SDPA said that in order to meet the iron ore goal expansion, it will be necessary to provide a capacity of 57 million gross tons by 1955.

Norway Finds Niobium Ore Extraction Method

The Norwegian Government mining organization, Norsk Bergwerk AS, has announced a new method has been found for processing the niobium ore recently discovered in Telemark, in northern Norway.

A pilot plant to treat about 40,000 tons of ore per year, yielding 150 tons of concentrates with a 50 pct metal content, is under construction. The U. S. has a fixed price for niobium, and it is expected that profits will be good. According to the director of the Norwegian Government group, demand in the U. S. is about 20 times the current supply.

Denver "Sub-A" Flotation

Denver Cross-Flow Classifiers

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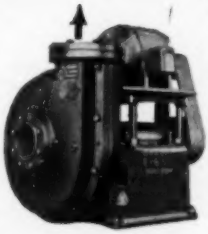

Examples of DENVER SRL Rubber Lined Pump Curves*

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		HP	.60	1.5	2.8	3.7	
3"x3" SRL	100	RPM	760	1053	1303	1453	
		HP	1.1	1.9	3.4	4.3	
5"x5" SRL	300	RPM	590	800	956	1087	
		HP	2.4	5.4	8.3	11.5	
6"x6" SRL	1000	RPM		862	1005	1122	
		HP		14.4	22.6	30.0	
3"x3" SRL-C	150	RPM	870	1145	1385	1580	1745
		HP	1.5	3.2	5.5	7.2	9.6
5"x4" SRL-C	350	RPM	655	850	1020	1160	1280
		HP	2.9	5.4	8.3	11.4	14.5
8"x6" SRL-C	800	RPM	500	635	780	890	980
		HP	5.7	11.6	16.8	22.3	28.6
10"x8" SRL-C	2000	RPM	485	610	710	808	855
		HP	14.0	27.8	41.2	56.3	71.6

(*Multiply these horsepower ratings by the specific gravity of your pulp to determine actual break horsepower required.)

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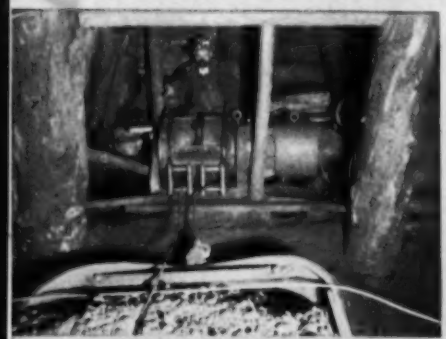
JOY AAF-211 *double-drum slusher in a lead-zinc mine. Mounted on a turntable, scrapes alternately from two headings.*

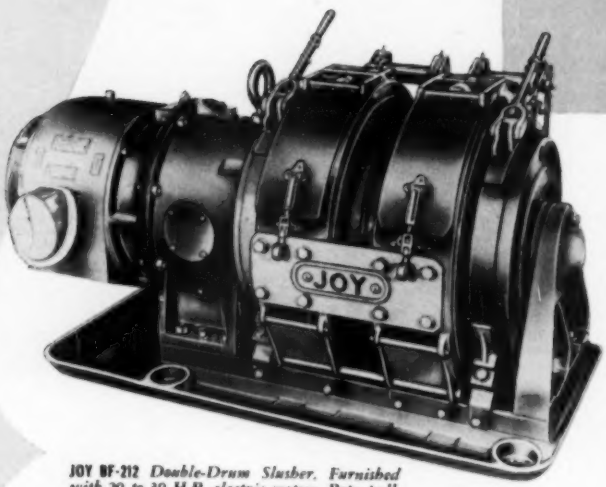
JOY AF-311 *three-drum slusher in a large open stope in a western copper mine.*

JOY AAF-211 *double-drum slusher in a cut-and-fill stope in a copper mine.*

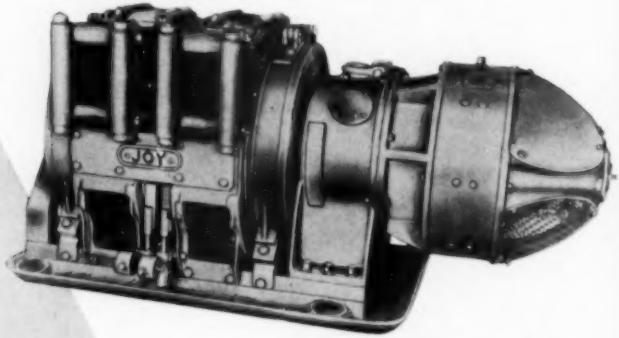


JOY BF-211 *double-drum slusher pulling ore to a chute in a Michigan iron ore mine.*

Here's why



JOY BF-212 *Double-Drum Slusher. Furnished with 20 to 30 H.P. electric motors. Rope pull, with drum half-full of rope, up to 3700 lbs.*



JOY CF-211 *Double-Drum Slusher. 30 to 60 H.P. electric motors. Rope pull up to 6800 lbs., with drum half full of rope.*

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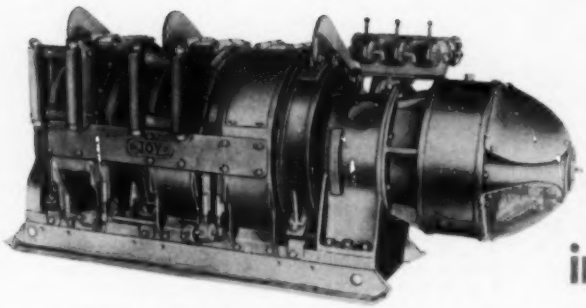

JOY CF-312 *Three-Drum Slusher equipped with air-clutch control for ease of operation. Electric motors from 30 to 60 H.P. Rope pull, with drum half-full of rope, is up to 6600 lbs.*

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- ★ The power unit can be easily removed, transported to the shop and overhauled without bringing the slusher above ground.
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THREE years ago, Russia virtually stopped all manganese shipments to the U. S., a nation dependent upon imports for about 90 pct of its supply of the metal. Russia was the chief source of manganese at that time and the loss seemed almost irreplaceable. Since then, however, developments in Africa, Asia, and South America have almost wiped out the effect of the Russian move.

India is now the biggest source of supply, with the Union of South Africa running second, and Latin America reviving its dormant manganese mining industry. In 1950, the U. S. was able to import 865,000 tons, a figure which declined only slightly in 1951.

Swift action on the part of the U. S. in 1948 and 1949 brought a rapid increase of manganese shipments from the Union of South Africa. The U. S. supplied urgently-needed steel for construction of ore cars used for mine shipments. South Africa's position in the supply picture is even more significant when it is realized that 15 years ago she exported no manganese to the U. S.

American steel companies have been instrumental in developing new sources. Latin America supplied more than half of the manganese shipped to the U. S. during 1944 and 1945, but since then production has fallen off badly. Private U. S. funds are being poured into finding and development of new sources in Cuba, Brazil and elsewhere.

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ALUMINUM Ltd., through its wholly-owned French concern, Bauxites du Midi, has started mining bauxite ore on the Island of Kassa, in the Los Archipelago, off the coast of French Guinea, French West Africa. The first shipment from the mine, the largest aluminum ore operation in Africa, is on its way to the company's smelter at Arvida, Quebec. The first shipment of 9804 long tons starts a shipping schedule of approximately 10,000 tons every 10 days. About 50,000 tons will be shipped during the rest of 1952, with a total of 300,000 tons expected in 1953.

The African operation took more than two years to complete with more than 1200 natives, and 100 Americans, Canadians and Frenchmen on the job. Aluminum's bauxite plant is the first industrial development in French Guinea, and as such posed problems which at first might have appeared unsolvable. Native workers are from the Soussou, Baga, Malinke, and Guerze tribes, living in villages of thatched straw and mud. Screwdrivers, hammers, chisels, and saws were as foreign to them as subway trains or ice cream sodas. It meant that a labor force had to be built from the rawest possible material, with instructional aides in written form almost completely ruled out. Even hod carrying provided a crisis. The natives refused to carry cement any other way but on top of their heads in baskets and buckets.

One modern device, however, has been adopted with enthusiasm. Natives drive all the trucks on Kassa. With construction at an end, most of the Europeans, Canadians, and Americans will go home. A labor force of about 300 natives will be administrated by 45 Europeans. Among the things that had to be done was the ridding of the island of tropical diseases. Malaria-carrying mosquitoes have been virtually eliminated and a small hospital established. The natives use the medical facilities, but hedge a little by consulting with their witch doctors.

Kassa ore is derived from a syenite bedrock. After blasting, the ore is loaded by power shovel and hauled by truck to crushers. Wide rubber belt conveyors carry the ore from one beneficiation stage to another. After kiln drying, the ore is stockpiled in a building with a 35,000-ton capacity.

Schools for native children, a village for European employees, a movie theatre, and artificial beaches have already been constructed. In addition to the on-the-job training at the aluminum plant, Island Governor P. H. Siriex has developed a four year plan, to start in 1953, which will include a program for the development of craftsmen and a definite progression of industrial development.

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LAST month, Aluminum Co. of America announced that it would build a gigantic aluminum reduction plant in Alaska. The project was contingent on certain conditions. In the excitement generated by the huge proportions of the undertaking, certain factors were overlooked. Primary reason for construction of the plant is the availability of Canadian water power. Canada seemed more than willing to grant water power rights to the U. S. firm. The water power was to be tunneled from the Canadian side of the border, to Alaskan hydroelectric plants for use by Alcoa. But some Canadians don't see it quite that way.

They see the power lines carrying Canada's burgeoning aluminum industry away from them and right into the hands of competitors. Aluminum Co. of Canada is racing to finish its Kitimat reduction plant. The first unit will cost an estimated $160 million. The total expansion for the plant will run about $500 million, with a 500,000 ton per year output when completed. The Alaskan project will eventually be able to turn out an estimated 400,000 tons.

Some Canadians see this as a definite danger. They see the plant using Canadian water power to compete with Canadian industry. Obviously, if Canada decides to withhold water power rights the Alaskan projects may never get beyond the planning stage. Certainly, Canadians can be expected to do all they can to protect their own interests. They see a great future for Canadian aluminum on the world market. Much of their industrial expansion is due to aluminum—and along with it, much of their new found wealth.

If Canada should decide that the Alaskan project will nullify the outlays for Kitimat, Aluminum Co. of America will have to find its water power somewhere else. In that event, Alaska will be the loser.

PHILIP D. BLOCK, JR., vice president of Inland Steel Co., visualizes a fine future for the Menominee Range of the Upper Peninsula of Michigan. He sees a future unclouded by threat from foreign ore imports or the development of low grade ore processing in Minnesota. The statement was made during the opening of Inland's new Cayia mine near Crystal Falls.

Inland, according to the vice president, is planning to use an increased amount of the high phosphorous ores found on the Menominee Range. He said that other steel companies will probably do the same, because the steel industry has learned how to use ores of high phosphorous content. He said:

"I do not mean to imply that we can use an unlimited amount of high phosphorous ores or that we can burden our blast furnaces with 100 pct high phosphorous ores, but we have reached the point where we can absorb a much larger percentage of these ores without any detrimental effects.

Foreign ores and lowgrade Minnesota ores cost too much to offer real competition, Block said. Transportation costs from the Menominee range to steel producing centers in Chicago and on Lake Erie are less than for any other range in this country.

In the meantime, the American operators of the Bomi Hills iron ore mine in Liberia, have reportedly signed a new agreement with the Government.

Several months ago, President Tubman of Liberia expressed a definite dissatisfaction with the original contract. Informants state that the agreement only needs the approval of the Export-Import Bank, which loaned Liberian Mining Co. $4 million to develop the iron ore properties. Majority stockholder of the Liberian Mining is Republic steel, with 61.57 pct of the stock.

Under the new contract, the Liberian Government will start to share in the profits when the indebtedness is liquidated and the return of invested capital accomplished, or April 1, 1957, whichever is earlier. Under the old agreement the Liberian Government received a 5 cent per ton royalty and an additional sum based on the price of pig iron. The royalty rate reached more than 17 cents per ton this year.

After debt liquidation the Liberian Government will receive 25 pct of net profits for the first five years; 35 pct for the next ten years; and 50 pct thereafter. Until the Government starts to share in profits it will receive a royalty of $1.50 per ton in U. S. currency, in addition to royalties provided for in the old agreement.

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THE hundredth anniversary of the North of England Institute of Mining and Mechanical Engineers was celebrated recently. The Institute is the oldest of the mining institutes which federated to form the Institution of Mining Engineers.

The Institute was originally chartered by Queen Victoria on Nov. 28, 1876. A brochure has been published tracing the history of mining development in the British Isles and the part played by the Institute in technical advancement. British mining can be accurately traced from the 13th century bell mining methods to the bord and pillar techniques of about 1740 to present day shaft sinking and ventilation techniques. The brochure also recalls that early members of the Institute were largely responsible for the foundation of Kings College.

The Institute maintains a wonderful library at Neville Hall, Newcastle-upon-Tyne, where a special meeting was held to celebrate the centennial. An exhibition of mining was also held at the University School of Mines, Kings College, many of whose initial undertakings began at Neville Hall. But like many similar organizations in the world today, the Institute is in financial trouble.

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THE Russian *Literary Gazette* recently carried a letter from a mining engineer's wife which told of working conditions which forced miners to sleep in her husband's office, hysterics on the part of the management, and general disorganization. The lady said her husband has been forced to snatch sleep while standing against a post underground, and rarely is able to sit down to dinner without being called back to the mine for no reason. Going to the movies with one's family means a public reprimand over the intercom system. The letter went on to describe other conditions equally as unbelievable. The *Literary Gazette* editors reported that they investigated and found the charges to be true. As usual, when Russian publications carry charges of this type, they have a double meaning. The letter probably was written, but perhaps for other reasons than appear on the surface. It just might be that someone higher up decided a new manager was needed. A letter, a witness, an investigation and someone takes a one-way ticket to Russia's new edition of the Kzar's salt mines—the uranium mine.

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AN Associated Press reporter, after traveling over a mine access road in the vicinity of the lead-zinc mines around Silverton, Colo., called it "a narrow thread of the modern world twisting up a gulch, knifing across a mountain meadow, hanging on the edge of a cliff and finally zig-zagging over a saddle between peaks in the chill, windswept world above the timberline." At first glance the words are the purple prose of a neophite in the mining world. Yet, it might be that familiarity with hardship inures one to the rough, tough life that many mining men are forced to lead daily. The AP man may have caught something most people never get to see.

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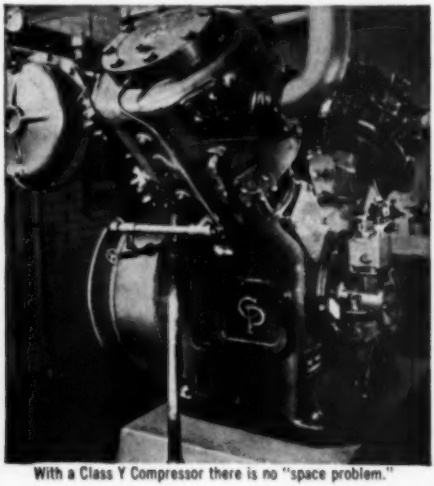

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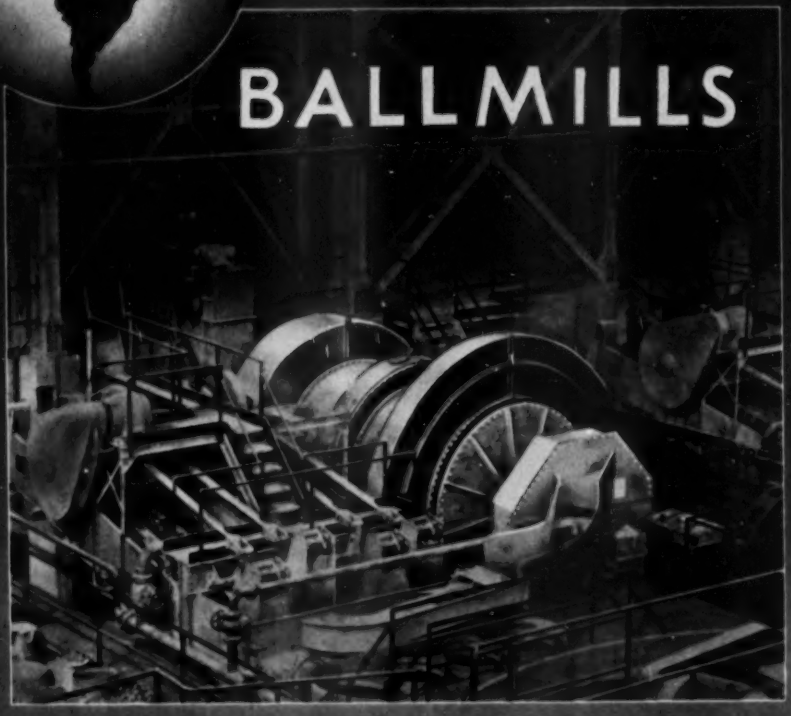

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MINING
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EDITORIAL

LAWN SPRINKLING AND POLITICS

WE vote this month without knowledge of either major presidential candidates' position on mineral problems. Too long have we kept our own counsel, turned our backs on the public. Certainly we must formulate policies upon which we may unite in industry council, but implementation of these by legislative bodies requires popular understanding, if not support.

The earth and its resources are the province of the miner and the farmer but beyond this, and the basic need of the economy for both, the similarity ends. In this election year the disparity between the poll strength of these two fundamental callings is apparent. While both candidates have found it essential to incorporate a farm plank in their platforms, mining has been neglected as not being a basic issue. If this gives the mining fraternity the advantage of voting objectively, it also leaves the general public in the dark as to the attitude of the candidate on a basic issue. It is an issue on which depends the future of the nation, but, unfortunately, the public is not aware of its own vital stake in mining.

Thus election time brings with it a fresh example of the evils of automatic lawn sprinkling—a long standing peeve of ours. Back in the days of the hand-held sprinkling hose in the intimacy of the twilight the mining man had the opportunity to unburden himself on the problems of mining to his neighbor who might be a doctor, lawyer, or accountant. With the encroachment of mechanized sprinkling, yet another forum for the indoctrination of the layman into the significance of mining has disappeared.

Familiarizing your neighbors with mining in the lawn sprinkling societies of yesteryear was the individual's contribution to public relations for mining. The corporations can use educational advertising in popular magazines as is practiced by such companies as Kennecott. The AIME can play an important part by providing career guidance information on the mineral industries. We offer the suggestion to the American Mining Congress, that the splendid exhibitions of mining machinery featured in alternate years for metal and coal mining, respectively, might be opened to the general public one night during the week of the show. Newspaper publicity would add greatly to the public's interest in such an opportunity.

Power for the public is considered a vote-getter and no candidate has neglected to make his stand on this subject. Is the price of the kilowatt in the home of greater importance to the voter than the price of the metal durable goods which he purchases every year? Not really, but the voter thinks so. Mining has as much sales appeal as potatoes or kilowatts if presented in the proper way.

We suggest that a little hand lawn sprinkling by the right people in the right places would produce an abundance of publicity which would give Mr. and Mrs. American something new to expect from their representatives—a vigilance over the flowering of their mining industry.

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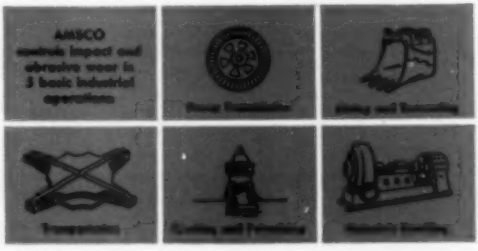

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Uranium Deposits of the Grants District, New Mexico

by Charles C. Towle and Irving Rapaport

URANIUM mineralization along the north flank of the Zuni Uplift, in the vicinity of Haystack Butte, was discovered by Paddy Martinez, a Navajo Indian, in the spring of 1950. The find was reported to the Santa Fe Railway Co., which initiated a program of exploration upon its holdings in the district in December of 1950. The Anaconda Copper Mining Co. secured numerous leases in January and February, 1951, and began an intensive exploration and development program in March 1951. The district was extended into the Lucero Uplift in August of 1951 by the discovery of uranium mineralization in Todilto limestone by Joy Sinyella, a Supai Indian living on the Laguna Reservation.

Initial reconnaissance was performed in the district by the Atomic Energy Commission in November 1950. A field office under the direction of the Denver Exploration Branch of the AEC was established at Prewitt, N. Mex. in January 1951 to study regional and detailed guides to ore, to evaluate the ore deposits, and to promote the development of the district by providing geologic services for the companies and individuals engaged in exploration.

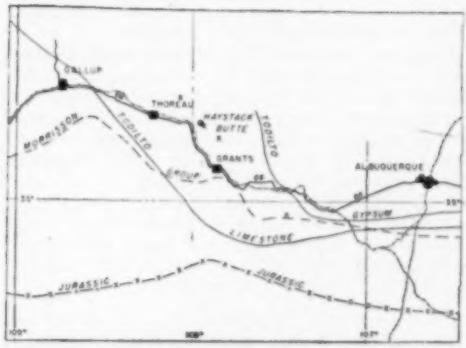
With the development of sufficient ore to warrant construction of a mill, the Grants area of New Mexico has come of age as a full fledged uranium mining district. The carnotite-type deposits of Grants, the first of major economic importance found in limestone, have extended the Colorado Plateau carnotite field into the more complex structure of the Basin and Range Province.

General Geology

The Grants uranium district is in northwestern New Mexico, along the southern flank of the San Juan Basin, between the cities of Gallup and Albuquerque. The region is at a mean altitude of about 6800 ft, vegetation is sparse, and outcrops are excellent. Mt. Taylor volcanic peak, at an elevation of 11,400 ft dominates the surrounding area.

The overlying cliff-forming Dakota sandstone is the most resistant formation in the Grants District and characteristically erodes into dissected mesas as much as five miles wide. The principle uranium ore-bearing horizons are found in the underlying Morrison formation, and the Todilto limestone of the San Rafael group. The Morrison weathers into a series of steep slopes and precipitous cliffs near the lower part of the Dakota-capped rims, while in contrast, the relatively resistant underlying Todilto

C. C. TOWLE is Chief, Denver Exploration Branch, Division of Raw Materials, AEC, and IRVING RAPAPORT is now with Running Hanosh Mines, Inc. at Grants, N. Mex.



Map of Grants District, where Anaconda is building beneficiation plant for uranium ores.

limestone forms broad benches extending out from the base of these Dakota rims as much as five miles wide. The Todilto benches are covered with overburden to an average depth of about 5 ft in the first thousand ft away from the Dakota rim, but increases to about 200 ft at the base of the rim. In many places however, a thick mantle of upper Cretaceous sediments and Tertiary volcanics obscures the ore-bearing Jurassic, and precludes exploration. The known uranium deposits of the Grants District are distributed throughout a zone 85 miles long and about 10 miles wide.

Stratigraphy

Rocks from pre-Cambrian through Tertiary are exposed in the uplifted areas, but uranium appears to be restricted chiefly to two zones in Jurassic sediments. The Jurassic sediments of the Colorado Plateau have been subdivided into three units, in ascending order: (1) Glen Canyon group, (2) San Rafael group, (3) Morrison formation. The Todilto limestone zone within the San Rafael group has produced the bulk of ore to date, but the promise of the Morrison zone is rapidly increasing.

The Todilto formation extends about 300 miles east-west and about 100 miles in a north-south direction. It is composed of a lower, extensive, limestone member and a central, more restricted, thick, pure gypsum member. The Todilto is believed to be of lacustrine origin.

The Morrison formation in this area is subdivided, rather arbitrarily, into three members: Recapture Creek, Westwater Canyon, and Brushy Basin. Ore

Rims exposed like this speed prospecting.

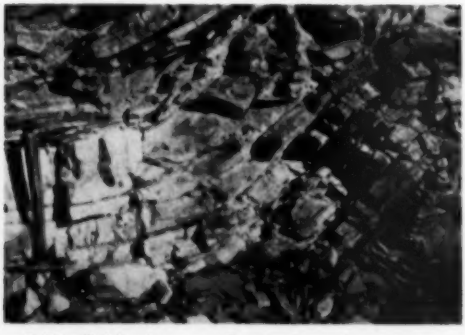
Anaconda field sampling plant, Grants, N. Mex.

is found in all three members but appears to be localized in the coarser clastics. Logs and twigs and an infinite variety of superimposed channels are characteristic of this formation. The upper portion of the Morrison formation is believed to have received considerable pyroclastic material.

The most significant regional information gained from the study regarding the tectonics of Jurassic sedimentation is the fact that the basin of deposition appears to have remained essentially constant throughout the Jurassic. Variations in sedimentation were chiefly the result of climatic change. Toward the close of the Jurassic uplift of the source area, the Zuni Highland, erosion was accentuated, resulting in the contribution of coarser clastics. This regional uplift finally culminated during lower Cretaceous and resulted in the progressive truncation of Jurassic sediments to the south.

Structure

The district is divided into three major structural elements: the Zuni Uplift which trends N. 60° E. over an area of 2000 sq miles; the McCarty Syncline, which covers approximately 1500 sq miles, and the Lucero Uplift near Laguna. The Zuni Uplift belongs to the southeastern portion of the Colorado Plateau geomorphologic province and abuts the Basin and Range Province to the east and south. The eastern part of the Zuni Uplift is believed transitional between the two physiographic types. It is complicated by a myriad of faults, and displays the effects of igneous activity in a series of extrusive rocks, of both fissure and cone types, which range in age from late Miocene to Recent. The following general conclusions regarding the structure of the Zuni Uplift have been advanced by Dr. Walter H. Bucher and Mr. Arthur K. Gilkey of Columbia University, under contract to the AEC: 1—Diastrophism has been active through the Tertiary, but appears to have been most intense during the lower Miocene. 2—The Zuni Uplift is chiefly the result of elevation, which is believed to be due to the injection of an early Tertiary subjacent pluton. 3—Faults and monoclines appear to be the expression of tensional stress. Joints are almost always tensional failures normal to the direction of most of the bending and normal to the bedding. 4—Major fractures in the pre-Cambrian, including those mineralized by fluorite, barite, pyrite, and copper, conform with the joints in adjacent sediments. 5—Faults appear to grade upward and laterally into fracture zones and monoclines.



Roll in Todolito limestone. This has been the most productive formation.

Distribution of Ore Deposits

Mineralization is confined to two stratigraphic horizons: the Todilto limestone and the immediately overlying calcareous silts and limestone lenses of the Summerville formation, and the Morrison formation. There is a stratigraphic separation of 500 to 800 ft between these ore-bearing zones. Ore deposits in the lower zone, the Todilto limestone, have been discovered in a zone starting from a point 2.5 miles north of Thoreau and continuing 65 miles southeast to a point 7 miles southwest of Correo. The easternmost known deposit in the Zuni Uplift is 30 miles distant from the westernmost deposit discovered in the Lucero Uplift across the prohibitively deep cover of the McCarty Syncline. Known mineralization in the Morrison formation extends from a point two miles northeast of Gallup for a distance of 85 miles southeastward to a point six miles northeast of Laguna. The sandstone deposits of the Zuni Uplift are 30 miles distant from the westernmost known deposits of the Lucero Uplift. The upper Morrison is exposed along the flanks of the intervening McCarty Syncline. The Morrison ore deposits are apparently smaller but richer than those in the Todilto, and are distributed over a stratigraphic range of 300 ft while the ore-bearing horizon in the Todilto limestone is confined to a total thickness of 16 ft.

It appears that the ore bodies, in their present form and distribution, are the result of the lateral migration of slightly heated Tertiary waters. The paragenesis and the sequence of geologic events deduced from field examination are believed to be:

1—Deposition of the Todilto limestone.

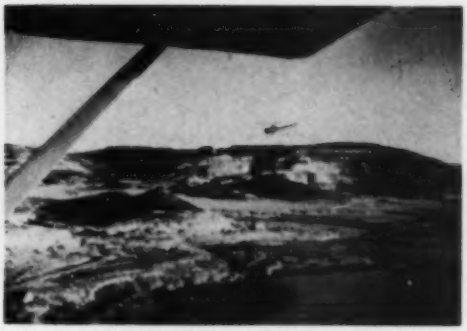
2—Recrystallization of the upper Todilto limestone and possible introduction of uranium.

3—Zuni Uplift with fracturing and faulting.

4—Introduction of coarse-grained black calcite (black color caused by presence of nontronite) along bedding planes and fractures.

5—Vein filling by coarsely crystalline white calcite.

Half-track mounted percussion drill provides mobility for exploration. Closely spaced jackhammer holes supplanted test-pits in some areas as experience was gained.



Looking north from Laguna, N. Mex., helicopter shows against sky. Airborne radiometric reconnaissance pinpointed promising areas for the prospector on foot.

6—Replacement of coarsely crystalline calcite by barite, pyrite, and fluorite.

7—Lateral percolation of uranium-bearing solutions through the Todilto. Uranium may also have been introduced directly from magmatic sources at this time.

8—Oxidation of pyrite to hematite, pitchblende to yellow uranium minerals, and the downward leaching of uranium minerals along joints.

Mineralogy

Uranium minerals identified in the Todilto limestone include the uranium vanadates, carnotite and tyuyamunite; the uranium silicates uranophane, beta-uranotil, and sklodowskite; and the uranium oxide pitchblende. The uranium vanadate and silicate minerals form the bulk of developed ore, but an increasing percentage of pitchblende is being found in deposits removed from the effects of superficial oxidation. Uranophane occurs chiefly along fractures and in cavities partially filled by calcite and barite. Gangue minerals, which occur in minute quantities, include hematite, pyrite, barite, traces of purple fluorite, and coarsely crystalline calcite. Only carnotite and schroeckingerite (hydrous uranium carbonate-sulphate) have been observed in the Morrison ore deposits.

Ore Controls

Ore controls discovered thus far have proved to be sedimentary structures and stratigraphic features. Mineralization transects folds developed during Zuni diastrophism, and neither faults nor joints exhibit abnormal radio-activity or hydrothermal alteration. There is a suggestion that broad, shallow, primary troughs in the Todilto have controlled the passage of ore solutions. Minor folds, the apparent result of Tertiary diastrophism, continue to be the chief structural guides to ore. The entire thickness of the Todilto (8 to 16 ft) may be mineralized on these folds, whereas generally, the adjacent flat-lying limestone contains uranium only in the upper, coarse-grained, recrystallized portion of the Todilto. In the Morrison, uranium solutions that migrated through sandstone appear to have travelled along local unconformities and the uranium minerals were precipitated in the presence of organic material.

Exploration Methods

Initial exploration by the Santa Fe consisted of sinking 6x6 foot test pits on a grid pattern of 100 ft centers, through the Todilto limestone. Bulk samples taken at intervals of 2 ft were crushed, sampled, assayed, and the lithology recorded. The outcropping rims of the Todilto benches were drilled and blasted to determine the magnitude of surface leaching. These methods were necessary since almost nothing was then known about the deposits. After one large ore body had been partly developed by this method, jackhammer drilling, adopting a 2 ft sample interval, was substituted for the initial test-pitting. Any test pits that were subsequently sunk around jackhammer holes have been sampled and assayed. Comparable samples taken by the two methods have shown an average variation of 62 pct; the discrepancy is believed to be chiefly due to the erratic nature of the ore. Santa Fe now uses a half-track mounted percussion drill in its exploration. Although a 100-ft grid pattern is still employed in areas where the depth of hole averages 40 ft, a wider spacing is contemplated for exploration of areas under greater depth. The maximum depth of percussion-type drilling has been 168 ft.

The Anaconda has employed a diversity of methods in its exploration, including bulldozing, trenching, and diamond drilling, in addition to wagon drilling and test-pitting. Pipeline diggers and a back-hoe were used on the broad Todilto benches to expose the top of the limestone. Bulldozing has been performed in exposing the ore-bearing zone on steep talus-littered slopes in the Morrison sandstone. However, it may be practical to explore the Morri-

Lightweight jackleg drilling equipment finds favor for small underground operations on irregular orebodies.

Shovel equipped tractor casts off bench in Grants District surface uranium mining operation.

son over a large area through the Dakota cap by drilling when better geologic criteria are developed.

Diamond drilling costs have averaged about $2.00 per ft, compared with approximately $0.75 per ft for wagon drilling. Below 60 ft wagon drilling costs rapidly approach those of diamond drilling. Although the greater depth limits and representativeness of the sample favor the use of diamond drills for deep exploration, the wagon drill is ideally suited for cheap exploration in favorable areas of shallow drilling on the Todilto benches.

Airborne radiometric reconnaissance by the Anaconda Copper Mining Company and the Atomic Energy Commission has been successful. Considering the thousands of miles of rim exposure in the Grants District, the impracticability of prospecting all of this large area on foot is apparent. Scintillometer readings above background are recorded on an aerial map, and these anomalies are then checked by ground reconnaissance. Several important discoveries have been made in this manner and the area of favorable ground reconnaissance has thereby been delimited. Refinement of this principle, using more sensitive scintillation equipment and automatic recording devices, should increase the safety in flying and the rapidity and accuracy of this method of radiometric surveying.

Mining Development

Maintenance of grade has been a major mining problem throughout the carnotite field. The average grade of the ore deposits in limestone of the Grants District is somewhat lower than that of Colorado Plateau ore. However, cheaper exploration and mining costs may compensate for the lesser value of the ore. Ore is erratically distributed in both plan and section, and rigid controls must be enforced to maintain the assay cut-off during mining. It may be necessary to sample drill holes 5 foot or less from the face; each dipper of ore may be tested at the pit by the use of a Geiger counter to determine whether it is ore or waste.

Experimental open-pit mining by Anaconda established the fact that ore transacts the bedding. As it is not possible to break cleanly to the vertical limits of the ore, 2 ft of basal dilution should be included in ore calculations. Some of the overlying barren silt and limestone may also be expected to contaminate the ore. Ore bodies of less than 100 ft in diameter may be mined, but one of the major mistakes being made in the area today is the purchase of too much large equipment for the small size of the ore bodies that are being developed.

Haulage is a relatively simple problem in the Grants area. Access roads to any of the known limestone deposits can be levelled by bulldozer, and the ore deposits are within about five to ten miles of U. S. Highway No. 66 and the main line of the Santa Fe Railway.

Mining Methods

Virtually all of the ore deposits blocked out on the Todilto benches can be mined by open-pit methods. The cost of stripping the ore is estimated at $0.50 per yd. This cost is higher than would normally be expected, but the small size of the individual deposits and the corrugated nature of the top of the limestone present additional complications. Actual mining including the cost of stripping and detailed sampling, will vary considerably depending on the characteristics of individual ore lenses.

Underground mining in the Morrison and in a few Todilto ore bodies presents a different set of problems. In general, the small size and irregular distribution of the ore in many of the deposits does not warrant a large capital investment. Small operations on the Plateau usually follow the ore wherever it leads, using a jackhammer on jackleg, hand-mucking, and wheel-barrow, but when a group of mines or a large mine is owned by a single company, mechanized methods may be employed. Some of the larger companies in the carnotite field are using two drill jumbos which are moved from mine to mine during the shift. Mucking machines, slushers, and diesel and belt haulage are also being put into use in larger operations. Small shovels, draglines, or crawler-type front-end loaders appear to be best suited for mucking limestone in these open pits. Rubber tired front-end loaders tend to spin their wheels while loading muck from an uneven floor.

The retreating long-wall method will be tested in the Grants District. Ground throughout the district appears to stand well without timber. It has been found that enriched uranium values are in the fines, and the possibility of producing a merchantable concentrate grade by dry crushing and screening is being investigated.

Ore Treatment

Arrangements have been made for the construction of an uranium mill and ore purchasing station at Bluewater, New Mexico, by the Anaconda Copper Mining Co., under a contract agreement with the AEC. The purchasing and stockpiling of both limestone and sandstone types of ore, was initiated on June 9, 1952.

Milling

Favorable results have been obtained in preliminary recovery tests on oxidized limestone ore from the district, but experiments on the beneficiation of uranium from pitchblende ores have not been equally successful; autoclaving is still required. It is believed that about 20 pct of the mill feed may consist of sandstone ore. However, if sandstone ore reaches major proportions, it may require initial roasting as the contained organic matter must be volatilized because of its interference with filtration. The milling process will consist of fine grinding followed by batch leaching in alkaline solution. The concentrate obtained in this manner is subsequently reduced and concentrated by smelting. Mill construction has been started, but it is anticipated that about one year will be required for its completion.

A typical top primed blast in an eastern Pennsylvania anthracite strip mine, displaying gas emanations and large quantities of fly rock. This gas and rock are products of the wasted energy of the explosive. The two pictures show the area prior to the blast and 1-1/3 sec after detonation.

Blasting — —

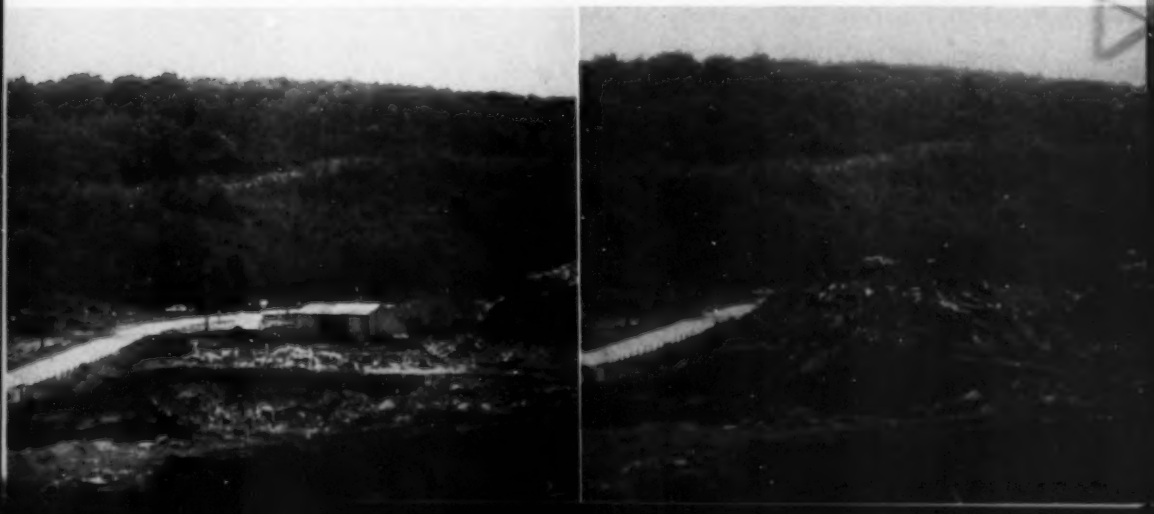
A BLAST can be "full of sound and fury," signifying nothing but a poorly confined charge, or it can be a muffled, well controlled explosion which moves the rock efficiently and places it in the desired location with the least amount of fuss and bother. Why one method will do the job and another proves to be almost useless is shown by the machine-gun camera studies of quarry blasting made by the Atlas Powder Co. staff.

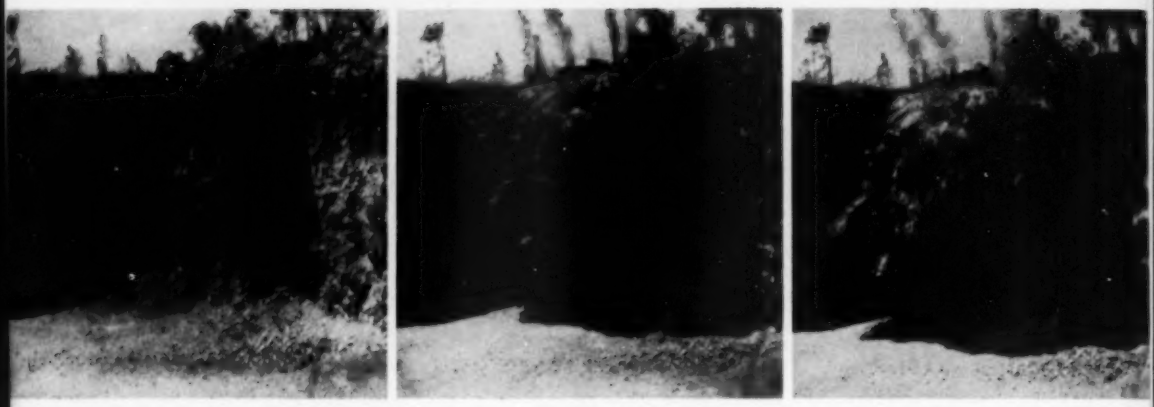
The sequence, or machine-gun, camera is the latest scientific device for the study of cause and effect factors in blasting. Movie cameras, previously used, although of some use, were limited because slow shutter speeds and small negatives failed to give clear, sharp prints. During World War II, the AAF developed the sequence camera, which figuratively stops the blast at regular intervals, making study and evaluation feasible. The camera is simple and inexpensive and offers material for accurate examination of blasting techniques.

The machine-gun camera takes three 5x5 in. exposures per sec at a 1/450 sec shutter speed, four times faster than a movie camera. Negatives are 30 times larger than those of movie cameras, giving large, sharp prints. The rock movement is frozen on the picture as the sequence of the blast takes place.

Pictures on this page show the advantage of individual machine gun photographs. But the sequences made possible by this camera give graphic, dramatic and conclusive support to recently developed detonation theories.

Controlled and effective blasting is within reach, as shown here in a coal strip mine. Here, with bottom priming, a successful blast was made adjoining a main highway and traffic was able to pass two minutes after firing. The two pictures show the area prior to the blast and 1-1/3 sec after detonation.





top primed time-0 sec | time-1/3 sec | time-2/3 sec

Camera Disproves Early Theories

Little consideration has been given to the location of the primer in the blast hole in quarry blasting, except from a safety viewpoint. The point of initiation was thought to have little effect on the blast because it was believed that a column of explosives detonated instantaneously. Placing caps in the bottom or top of a hole or using detonating fuse was supposed to give about the same fragmentation and rock movement.

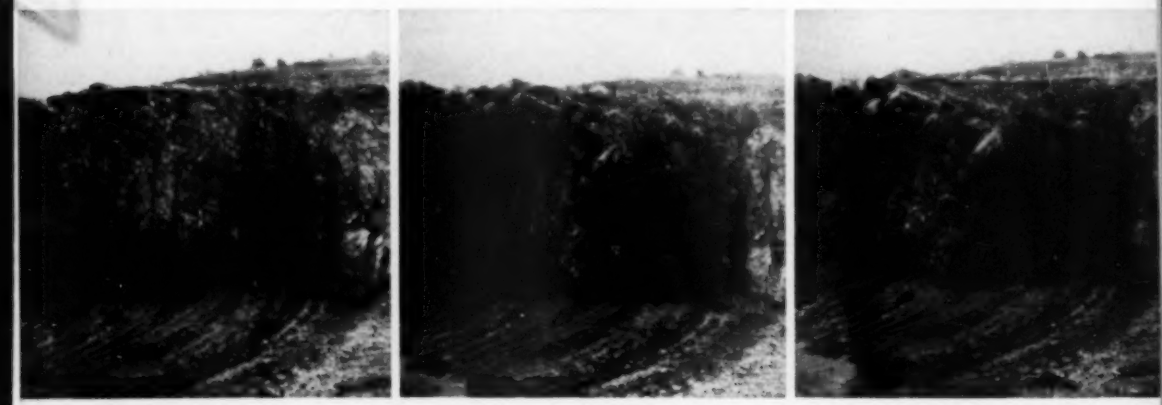
Sequence photography has made information available that disproves these concepts. It is obvious from examination of the series that a pronounced difference exists between top and bottom initiation.

Movement of Blast Material

Comparison of the two sequences shown points up definitely that rock movement starts at the point of initiation. In the blast primed at the bottom, note that the foot of the bank is ahead of the rock at the top, and maintains its lead throughout the blast. Lack of geysers at the top of the holes indicates proper confinement and utilization of all the useful energy of the explosive. Absence of flyrock is important as well as the fact that escaping gas is not noticeable until the fourth picture. Maximum explosion confinement and retarded gas escapement means maximum fragmentation and minimum noise and air blast.

On the other hand, examination of the blast primed at the top shows no movement at the bottom, in fact light can be seen under the falling rock until one second after detonation. Geysering appears in the first picture of the sequence, indicating energy losses at detonation of the blast. Large amounts of flyrock are produced by top priming and escaping

bottom primed time-1/3 sec | time-2/3 sec | time-1 sec



time - 1 sec

time - 1 1/3 sec

time-complete

gases appear at 2/3 sec, giving rise to the belief that a large quantity of the available energy was wasted.

Confining the blast helps avoid remnants or development of a toe. With bottom priming there is effectively only one free face at the point of initiation, and the face is outward, not upward. With top initiation there are two free faces, and the effect of the explosive tends to be dissipated upward. Geysers in the top initiated sequence show this latter condition.

In addition to economic advantages, in addition to getting the most work out of the least materials and labor, confinement of the blast reduces noise and shock. Noise reduction and minimum vibration are not only desirable in populated areas, but sometimes essential where one must work near existing buildings or roads. Better blasting means better public relations.

The pictures on page 1041 show the comparison in amount of outwardly visible blast effect when top priming and bottom priming; show the difference in the amount of flyrock. Cars were using the road in the lower picture sequence two minutes after the blast.

Safety Considerations

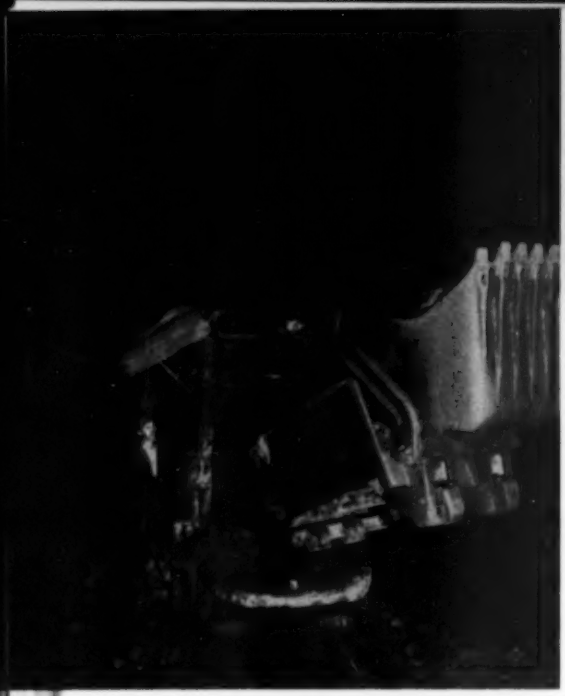
Bottom priming has met resistance because of the safety factors involved in different primer types. Firing in this manner requires standard or millisecond delay electric caps, and in the case of a misfire presents hazards in removing the primer cartridge. With top initiation, detonating fuse is used and is not as dangerous. However, dynamite manufacturers have recommended drilling the blast hole five feet below grade and priming the second cartridge. This places the blasting cap below the working floor of the shovel dipper.

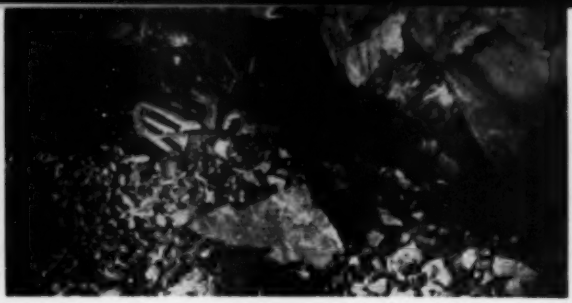
Continuing study of blasting using the camera technique will develop data on the effects of variation in timing, in type of dynamite, and in timing delay sequences.

time - 1 1/3 sec

time - 1 2/3 sec

time-complete





LEFT—Inside the tunnel at West Tahtsa Camp a Conway 100-hp rail-mounted mucker goes to work in the main tunnel heading. Lights in the background are for illumination for the cherry picker handling the narrow-gage cars transporting muck to the portal. ABOVE—Deep in the heart of the mountain an International TD-24 dozes rock blasted from the roof arch of the powerhouse cavern to a shaft. The material drops down the shaft into rubber tired dump trucks operating in the lower tunnel at the powerhouse floor level.

Hard Rock At Kitimat

ONE of the largest hard rock operations ever undertaken on the continent is part of the Aluminum Co. of Canada's gigantic British Columbia development in the mountainous wilderness back of the north coastal port of Prince Rupert.

To provide electric power for the huge aluminum smelter under construction at Kitimat, 400 miles northwest of Vancouver, Alcan is reversing the flow of a 5000 sq mile drainage area by blocking the eastward-flowing Nechako River with the largest sloping clay core dam in the world. The backed-up waters of a 110-mile chain of lakes will be tapped at their eastern end by a 10-mile tunnel piercing a mile-high mountain range. From this tunnel water will pass through penstocks to a subterranean powerhouse after a drop of 2600 ft and empty into the Kemano River almost at tide water.

A projected battery of eight turbine-driven generators will produce more than a million horsepower, although, initially, three generators of 140,000 hp each will be installed. Power will travel 50 miles across another mountain range, over a glacier, and across an arm of the ocean to what will ultimately be the world's largest aluminum smelter—a plant 2½ miles long.

Ore smelted at Kitimat will come from Jamaica, but carving the tunnel, penstocks, and powerhouses cavern out of the heart of a mountain is so vast a job that most of the problems of a major mining operation are encountered. The powerhouse cavern alone is a big hard rock job. This chamber lies 1400 ft back from the mountain face and will eventually be 1100 ft long, 80 ft wide, and an average 118 ft high. Now under construction, the roof of this cavern will take the form of a parabolic arch. The prime contractors, Morrison-Knudsen Co. of Canada, are to excavate a length of 700 ft by May 1954. This section will provide a main control room, erection bay and space for eight generating units.

The roof arch is to be lined with reinforced concrete and a plan now under study provides for concrete curtain walls separated from the rock sides of the cavern to prevent moisture accumulation.

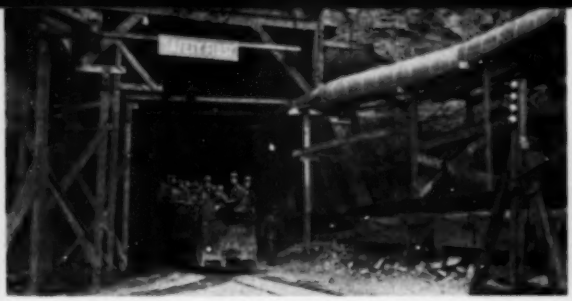
Cavern Excavation

Excavation of the cavern began with the driving of an 8x8-ft exploratory drift 1500 ft long. Two horseshoe-shaped access and tailrace tunnels were bored to serve excavation work. These tunnels are approximately the same length as the exploratory drift and 25 ft high. The floors of both tunnels were graveled to permit the use of rubber-tired equipment. Eventually, the tailrace tunnel will be enlarged to 40 ft height to drain water from the turbine pits below the powerhouse floor.

When the tunnels reached the cavern location, a mucking tunnel, approximately 25 ft square, was driven the projected length of the powerhouse at the floor level of the cavern. Another tunnel, 10 ft square, was carried in the same direction from the exploratory drift at the level of the arch spring line, 65 ft above the cavern floor. To connect these two tunnels, seven shafts 10 ft wide and approximately 80 ft long were driven through to the lower mucking tunnel, 65 ft below. These shafts were located on 120-ft centers, the length of the powerhouse cavern.

Blasting is done from this upper tunnel to the final line of the roof arch. Two bulldozers, an International TD-24 and a TD-9, doze blasted material to the shafts where it drops to floor level. From here, the rock is loaded by Eimco loaders and power shovels into rubber-tired Koehring Dumptors and Euclid end-dump trucks.

The arched roof section will be completely excavated and concreted before vertical blast holes are drilled down from the level of the arch spring line to the floor of the cavern. The arched roof section with its parabolic curve will have a width of 80 ft and a rise to the crown of 47 ft.



RIGHT—Hoisted half a mile up the mountain, a crawler tractor backs off the tramway at the Kemano end of the main tunnel at the 2600-ft level. The tractor was hauled up with only its blade removed, after remounting the blade the TD-24 went to work dozing muck removed from the tunnel. ABOVE—Change of shifts at the Horetzky Creek adit finds day crew leaving tunnel via locomotive and muck car. The large duct entering the tunnel provides ventilation, and the smaller pipe carries compressed air for drilling.

Mining Job Power Site

Excavation of all eight of the 9-ft manifold tunnels leading from the two main penstock raises into the turbine pits has been completed. Still to be excavated is the valve chamber which will form a lateral connection between the manifold tunnels and will house the valves between the pressure manifold tunnels and the penstocks. This chamber will be excavated by the same general long hole diamond drilling method used throughout the powerhouse cavern work.

Another large item on the unfinished list is the tunnel which will carry high voltage current to the transmission line. This tunnel will be 17½x13½ ft cross-section and approximately 1500 ft long.

Most of the smaller 9-ft tunnels were drilled with a mobile six man drill frame handled by crawler tractor. The balance of the small tunnels were drilled by Canadian Copco equipment mounted on jack legs.

Penstocks and Tunnels

Penstocks are being driven upward at 48° from the horizontal to meet the main bore which will bring water 10 miles through the mountain at the 2600-ft level. The two circular tunnels were driven horizontally for 500 ft before angling upward. Two more headings were started by means of a 100-ft adit at the 1685-ft level which branched off on each side to intersect the penstocks. Construction of the adit was accomplished with a jumbo frame from which nine Canadian Ingersoll-Rand DA-35 drifters were operated. At this level the penstocks again will be carried on a horizontal plane for a distance of 700 ft. At the foot of the slanting penstocks, gates are installed to load muck into Koehring Dumptors.

The penstock raises are excavated to 14-ft diam, but the final steel lining will be 11 ft. To provide strength to resist 1200 psi pressures rock will be grouted and the void between rock and inner steel liner will be filled with concrete.

The 10-mile, 25-ft diam main tunnel leading from West Tahtsa Lake west to Kemano, is being pushed in both directions from an adit at Horetzky Creek, halfway between the intake and the Kemano heading as well as from each end. Half a mile of tunnel is scheduled for completion per month.

Conway 100-hp rail mounted muckers with a capacity of 120 cu yd per hr handle rock at all headings. Drilling equipment at the Kemano and two Horetzky headings consists of 15 Canadian Ingersoll-Rand pneumatic feed drifters, with 5-ft feed range, handling standard steel with X-point detachable carbide bits. Drilling equipment operates from a three-deck jumbo. Crews at the heading for each shift are comprised of 15 drillers, 15 helpers and 3 to 5 additional men.

At the West Tahtsa bore crews of 16 drillers using Canadian Copco equipment with 5 helpers make up the drilling teams. Smaller crews are possible because the drills, using chisel-point carbide bits, are lightweight and require fewer chuck tenders than the conventional heavy drifters.

It has been necessary to reinforce the walls and sides of the tunnel at West Tahtsa since the rock consists of blocky porphyrys and andesites. For the first 1000 ft timber was used, and from there on steel sets were employed. All sections requiring support will be concreted and before this is done the wood sets will be replaced with steel.

A typical drilling cycle for all heading crews runs about 1¾ hr for drilling from 87 to 96 holes 13 to 15 ft deep. Loading takes ¾ hr and mucking takes the balance of the shift, 4½ to 5 hr.

Normal rate of advance is about 12 ft per shift. Forty eight-ft advance has been made over a three-shift period, but where the ground is poor the shift may not be able to complete the cycle of drilling, blasting, and mucking. In August the average rate of advance was 29 ft per day.

AIME Shares In Centennial Observance With Outstanding Technical Sessions

THE Annual Fall Meeting held in Chicago in conjunction with the celebration of a Centennial of Engineering brought forth a round of field trips and symposiums which did much to add to the store of information of the 240 who registered for the gathering. Instead of a scattered program, technical papers were grouped together in organized symposiums, where the particular importance of each report was enhanced by presentation in context.

Despite the heavy accent on technical papers and informative field trips, those attending the meeting had ample time to enjoy the benefits of a visit to Chicago. The AIME dinner dance was held at the Palmer House, in the Red Lacquer Room. The Chicago Section played host Friday evening at dinner.

The MBD held four sessions at the meeting headquarters, the Palmer House, September 4 and 5. Featuring the meetings was a review of taconite concentration and new theories on comminution. The first session opened with a paper on the *Structural Features Which Effect the Concentration of Iron Formations*, presented by F. J. Tolonen. The paper was centered upon the various members of the low grade iron formations of the separate Michigan districts. The study correlated macrostructure, microstructure, and concentration tests.

The executive committee of the Minerals Beneficiation Division exchange notes in an informal moment at the Chicago Fall Meeting. They are, from left to right: T. B. Counselman, J. F. Myers, E. H. Crabtree, W. B. Stephenson, Will Mitchell, and D. W. Scott.

Beneficiation Characteristics of Michigan Low Grade Iron Ores, presented by F. X. Tartaron, co-chairman of the session, reviewed the work of his laboratory on the beneficiation of low grade ore taken from various strata of a deep shaft on the Marquette Range. The paper included data on flotation, gravity separation, magnetic separation, and combinations of these processes. The economics of low grade ore treatment generated great interest.

Ty. Maki offered a compilation and correlation of field results with magnetic separators in ferrosilicon and magnetite medium recovery circuits.

Following the morning session, 130 persons attended a luncheon, followed by a brief business meeting. Later, a film illustrating the grinding, separation, and pelletizing steps investigated by the Mines Experiment Bureau, was presented by H. H. Wade. T. E. Ban, and L. J. Erck gave a report on pelletizing which evoked much discussion. R. A. Limons presented a paper on the *Mineralogy of Blast Furnace Sinter*, for its author, H. M. Kraner.

Friday sessions featured a new hypothesis for high speed drill rod milling and a new technical approach to comminution. Jack Myers commented that Consolidated's large rod mill was a hammer mill turned inside out. Steve Erickson described the use of *Cyclones for Heavy Media Concentration of Iron Ore Fines*. Bob Meaders gave a *Progress Report on the Aerofall Mill*. Shiou-Chuan Sun described an improved high frequency analyzer for studying the conductivity of suspensions of various minerals. L. A. Roe discussed a magnetic reflux condensor which enabled him to determine the degree of liberation of magnetic iron ores. Roy A. Johnson described a function of the hydroscilator in which the fan rake was eliminated in favor of a conical bottom.

The Industrial Minerals Division sessions held September 5 and 6 and were well attended. The Friday morning session was particularly concerned with developments and advances in industrial minerals with special emphasis on the Chicago area. The first two papers dealt with transportation on the Great Lakes and fuel movement into the Chicago area. The third paper proved to be of great interest, turning the spotlight on the fairly new hydraulic mining methods introduced by a silica company in the Ottawa, Ill. field.

M. L. Haider addressed the Industrial Minerals luncheon. At the head table, from left to right are: G. H. Waterman, S. S. Cole, E. H. Robie, R. C. Stephenson, M. L. Haider, and H. M. Bannerman.

Friday afternoon was given over to new developments in building materials in the Chicago area. Possibly the outstanding paper, according to those attending the Ind. Min. Div. sessions, was one dealing with new theories of multi-storied building construction embodying unusual designs in structural clay products. Lightweight and ultra-lightweight aggregates were covered in two papers discussing their general nature and particular usage in the Chicago building trades.

The symposium on Saturday covered recent developments in the role of groundwater in industry. The session was jointly held with the Geophysics subdivision and the American Association for the Advancement of Science. The symposium presented a *bird's* eye view of the latest *developments* in this field.

Coal Division sessions, September 5 and 6, were notably successful, with J. O. Osterberg of Northwestern University generating a great deal of interest with his paper, *What We Know About Slope Stability and Its Application to Strip Mining. Effects of Waterborne Industrial Wastes on Stream Pollution*, by Henry F. Hebley, Pittsburgh Consolidated Coal Co. received equal attention. Two other papers, *The Flow of Magnetite Suspensions Through Pipes*, by T. C. Fraser, University of Pittsburgh, and *Developments in Auger Mining*, by W. R. Hennessey, Cardox Corp., held the interest of those at the session. J. A. Simon and Clayton G. Ball discussed estimates of coal reserves in Illinois and the nation in general.

M. M. Leighton addresses the Geology session. Seated at the table are co-chairmen J. D. Forrester, on the left, and E. J. Clark. Leighton discussed the activities of the state geological surveys.

The Geology subdivision symposium on *Brown Iron Ore Resources*, held on Friday, saw papers presented by Walter B. Jones, Alabama state Geologist; Garland Peyton, director Georgia Dept. of Mines, Mining and Geology; Alfred E. Boerner, Geologist, M. A. Hanna Co., and George M. Schwartz, director, Minnesota Geological Survey; John T. Lonsdale, director, Texas Bureau of Economic Geology; Edward L. Clark and Garrett A. Muilenburg, Missouri Geological Survey; and Arthur J. Blair, consulting geologist.

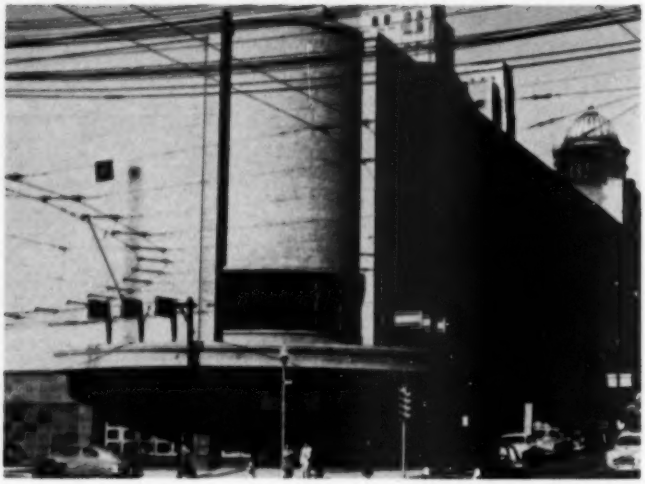
The Saturday morning session centered around the *Activities of the State Geological Surveys*, with George M. Schwartz; Franklin G. Pardee chief of the Michigan Geological Survey div.; M. M. Leighton, chief of the Illinois State Geological Survey; and Charles F. Deiss, Indiana state geologist, taking part.

The session continued after lunch. J. H. Melvin, div. chief of the Geological Survey, Department of Natural Resources, Ohio; E. F. Bean, Wisconsin state geologist; and Harold Bannerman, chief, div. of Economic Geology, U. S. Geological Survey, presented papers dealing with the symposium subject.

The Iron and Steel Division held a session with papers presented on *Developments in Steel Alloys*, by Elmer Gammeter, director of Globe Steel Tubes Co., and a Review of Wide Strip Rolling by D. S. Holstein, General superintendent, Rolling and Finishing, Middletown div., Armco Steel Corp.

Thursday September 4, the Illinois Geological Survey conducted a tour to the Ottawa and Goose Lake areas. Aside from a lengthy delay because of breakdown of the bus, the trip was an enjoyable one. Delegates saw the plant of the Standard Silica Corp., of Ottawa, where open pit and underground hydraulic silica mining from the St. Peter formation is under way. Second stop was at the Illinois Clay Products Co. plant at Goose Lake. The party was shown selective mining of refractory fire clay, illite bonding clay, manufacture of Zonolite insulating bricks and plaster, and the burning of ladle brick in the tunnel and beehive kilns.

The Indiana Geological Survey led a field trip to the Crisman Sand Co., The Hanley Sand Co., and the Bos Sand Co. A stop was also made at the Indiana Dune Sand State Park for study of the geological features of the Tolleston and Calumet dune and beach deposits.



Denver's new City Auditorium housed the huge four-day American Mining Congress equipment show, which attracted more than 5000 miners and suppliers. There were 146 exhibitors on hand to show off the latest in equipment and machinery.

Mining Show Attracts Record Crowd

MORE than 5000 miners and suppliers descended upon Denver to make the American Mining Congress' four-day metal mining show one of the most extravagant equipment displays ever assembled in one place at one time.

The most recent developments in equipment for mining, milling, and processing ore were shown by the 146 exhibitors at Denver's new City Auditorium which covers 57,000 sq ft of floor space. Other displays were placed in the open air opposite the auditorium site.

Registration desks were opened Sunday, Sept. 21. A field trip to the Colorado School of Mines at Golden was the featured activity of the day. Official opening of the show took place on Monday, and presided over by Howard I. Young, president of American Zinc, Lead & Smelting Co.; Otto Herres, vice president, Combined Metals Reduction Co.; William J. Coulter, vice president, Climax Molybdenum Co.; Charles B. Stainback, Westinghouse Electric Corp.; and Merrill E. Shoup, president, Golden Cycle Corp. Convention welcoming addresses were delivered by a representative of Mayor James Q. Newton, of Denver, and Gov. Daniel V. Thorton of Colorado.

Following the formal opening ceremonies a round table discussion dealt with the "Paley Commission Report" formally called *Resources for Freedom*. Participating in the panel were:

Arthur H. Bunker and George R. Brown, members of the Paley commission; H. I. Young, president of the American Mining Congress; Horace M. Albright, president of U. S. Potash Co.; Evan Just, vice president, Cyprus Mines Corp.; Donald H. McLaughlin, president, Homestake Mining Co.; E. H. Snyder, president Combined Metals Reduction Co.; Simon D. Strauss, vice president, American Smelting & Refining Co.; Sen. H. C. Dworshak (R.-Idaho); Ira B. Joralemon, consulting engineer; Jess Larson, Defense Materials Procurement Agency; and Andrew Fletcher, president of St. Joseph Lead Co.

The panel discussed the report for six hours, adding a great deal of information to the already voluminous amount which has been gathered concerning it. Three objections to the report frequently mentioned were: 1—the multilateral contract; 2—international buffer stocks; 3—international buffer stocks combined with limited quota arrangements. Points on which the panel commended the report were its recommendations for acceleration of topographic and geologic mapping of the United States and Alaska; recognition of the principle of percentage depletion for metals and minerals; and advocation of removal of the present limitations on the expensing of exploration costs applicable to minerals.

Technical sessions were held throughout the four days of the meeting. Included in the program were: *Progress in Mechanication; Small Mines; Public Lands; Open Pit Mining; Manpower-Labor Relations; Shaft Sinking; Roof Support; Taxation, Gold, Silver and Monetary Policy; Advances in Mill Practices; Special Conference on Strategic Materials; Uranium Mining-Atomic Energy; Prospecting and Exploration; New Metallurgical Processes; and Rock Breaking.* About 45 papers and several panels were presented during technical sessions.

Formal and informal social functions consumed the remaining time of the convention. Three locations were needed to handle the huge throng attending the Mining Jamboree Monday evening. The only other sponsored social function took place Thursday evening. A great many informal gatherings, chiefly convivial, were held in manufacturers' courtesy rooms—called wet stopes.

Field trips included Cripple Creek, Pueblo, Climax-Leadville-Gilman, Colorado Plateau-Grand Junction, Red Mountain, and Rifle-Rangley.

Mining Taxation

by Granville S. Borden

While pointing up the present status of mining taxation this article presents a definite program for progress and underlines "the incessant job" of maintaining present improvements in tax structure. On the following pages are also presented the opinions of experts expressed as answers to a poll sent by the author to executives of large metal mining companies.

MANY complaints have been filed by mine prospectors, explorers and producers about inequities in mining taxation. They have insisted upon improvement, and in the past 35 years there has been some progress. Many of the best taxation provisions made to date are those contained in the Revenue Act of 1951.

Mining economics presents special problems, a fact even recognized by many textbooks. Certain provisions have been made to take care of mining taxation rights, but these have been made by Congressional act and do not represent basic, constitutional, guarantees. Despite special provision, in the form of depletion allowances, there remain injustices. The industry faces the problem of gaining adequate income tax allowances for exploration and development expenditures, operating losses, and bonus and subsidy payments.

Problems in mine taxation arose soon after the adoption of the 16th Amendment. This amendment gave Congress the power to impose income taxes. Supreme Court decisions against a number of mining companies held that Congress, under its power to tax income, could tax net operating revenue before depletion allowance was deducted. The court also ruled that the right to deduct depletion is a matter of Congressional permission.

There is no statute provision permitting mining stockholders to recover their investment out of dividends while the mine is being operated. The Supreme Court did not recognize that part of mining proceeds are actually amortization of the cost of mineral rights. Both these omissions stem from failure to recognize that by its very nature mining consumes its mineral rights and therefore that a part of gross proceeds are not income but recovery of capital.

Suppose a Congress unfriendly to mining interests repeals the provisions that allow deductions for depletion. Under such a situation assume that a corporation buys a mineral deposit for $300,000. They mine the deposit and earn an operating profit of $500,000. At the current rates they would pay a Federal Income tax of in excess of $250,000. This venture results in a loss of $50,000 as shown in the table below. For the example given the absence of depletion provisions in income tax law would turn ore into waste rock, because the word "ore" connotes commercially mineable rock.

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Example of Mining Operation Without Depletion Allowance

	INCOME	EXPENSE
Purchase Price		$300,000
Operating Profit	$500,000	
Income Tax (Without Depletion Allowance)		250,000
	$500,000	$550,000
Net Loss		$ 50,000

In retrospect the conclusion is clear: If the Supreme Court had recognized that a part of mining proceeds represent in reality sales proceeds of mineral rights, then profits derived from such sales would have been taxed at the favorable long-term capital rates, instead of being taxed at the high normal, surtax and excess profits rates.

It is a question whether percentage depletion allowances offset this tax situation.

History

The history of depletion allowances has been one of general improvement. The original act did not include depletion, but an allowance at the rate of 5 pct of gross proceeds was provided in 1913. The Acts of 1916 and 1917 allowed depletion upon cost of property, and in 1918 the law was further changed to permit depletion upon cost, or market value as of date of discovery.

In 1932 metal, coal and sulphur properties were permitted to elect depletion in percentages of gross income, limited to 50 pct of net income, in lieu of cost depletion. Shortly thereafter the rule was changed and percentage depletion is now allowable in any case where it is higher than cost depletion.

Since 1942 percentage depletion allowances have been extended to cover many nonmetals, and in 1951 the rate for coal mines was increased from 5 to 10 pct of gross income. It is interesting to note that these allowances have been retained, and even

extended, over the past two decades despite attacks upon the provisions by the President and high-ranking Treasury officials.

Exploration and Development

For many years, the Bureau has prescribed through its regulations that "all expenditures in excess of net receipts from minerals sold shall be charged to capital account recoverable through depletion while the mine is in the development stage," and "the mine will have passed from development to a producing status when the major portion of mineral production is obtained from workings other than those open for the purpose of development, or when the principal activity of the mine becomes the production of developed ore rather than the development of additional ores for mining."

These rules were not stimulating to mining. Development and exploration expenditures incurred prior to production could not be deducted from income from other properties and these capitalized expenditures were classified as "capital recoverable through depletion."

By classifying exploration and development expenses as depletable rather than depreciable items, or deferred charges, the allowances for percentage depletion were substantially diminished. In most cases depletion allowable on cost, even with the addition of capitalized items, is still less than that allowable by percentage of income.

Arguments can be presented against this classification of development and related expenditures. The word "deplete" as used in Income Tax law connotes the exhaustion of a wasting natural resource through exploitation, not the loss of the useful value of capital invested in haulageways, adits, shafts and drifts. These are similar to transportation facilities and should be depreciable, not depletable assets.

The validity of regulations classifying development expenditures for oil and gas wells has been challenged before the Supreme Court. Taxpayers claimed that holes in the ground used to conduct oil to the surface are like a pipe line and as such are "depreciable", not "depletable" assets. The Supreme Court did not rule on this question; they sustained the regulations under a rule of statutory construction that any re-enactment of a statute without change constitutes tacit Congressional sanction of all existing regulations of the prior statute.

New Incentives

In 1951 Congress provided new incentives for mining by amending the Internal Revenue Code to improve deductions for development and exploration expenditures. The first change allows deduction of development expenditures, and the second change allows deduction of exploration expenditures up to $75,000. Under both provisions the taxpayer can elect to treat the expenditures as deferred charges deductible on a ratable basis.

The option to deduct expenditures currently or to defer the expenditures, and the right to keep the deferred charges out of the "depletable" category are important improvements. However, it should be borne in mind that the new laws apply only to expenditures incurred after January 1, 1951, and do nothing to change the rule for recovering adjusted bases for depletion or deferred charges for prior expenditures.

Loss of Capitalized Expenditures

In the past the Bureau ruled that abandonment or sale of the property upon which the work had been performed was prerequisite to deduction of capitalized exploration or development costs. The Bureau assumed that capitalized exploration and development were additional costs of mineral rights; that they were not separate assets. As they became embedded in the cost of mineral rights losses could not be taken even when facts indicated exploration and development knowledge to be worthless. The deduction must be deferred until the mineral rights are abandoned or sold.

There is some authority for the statement that the Bureau has modified this rule so that deductions can be taken from income of the year in which the taxpayer can prove the knowledge acquired from the exploratory work became worthless, even if title to the mineral rights is retained.

Improved Rulings

The Bureau of Internal Revenue minimized benefits flowing to taxpayers from percentage depletion by various rulings. Three of these regulations should be mentioned as having been corrected. Congress improved the definition of gross income, used in determining percentage depletion allowances, by including specific processes as part of mining.

The original formula allocating aggregate costs and profits between mining and metallurgical or other processes eliminated from mining profits substantial amounts emanating from fortuitous discoveries. The regulations requiring allocation on the basis of relative costs have now been amended to permit taxpayers to use more reasonable methods.

Earlier regulations excluded mine to plant transportation costs and profits from determination of gross income. Congress has since provided that transportation up to 50 miles, or longer hauls where shown necessary, must now be included in determination of gross income from property.

Excess Output and Excess Profits Tax

To provide critical and strategic minerals for World War II producers were urged to accelerate production rates. Compliance would cost the producer excess profits taxes which would not accrue if the deposits were mined at normal rates. To provide relief net income above that which would have been realized at normal output was exempted from excess profits tax, the proportion freed varying inversely with mine reserves.

For example a taxpayer with unlimited reserves would suffer no hardship from any acceleration of output. Notwithstanding this concept, the law allowed a deduction of 50 pct of the net income from excess output in the case of coal and iron mines. The excess profits Act of 1950 extended these World War II provisions, and the 50 pct rule, which had applied only to coal and iron, was extended to cover metal mines.

Under the excess profits tax laws of World War II, coal and iron mines which had no base period experience, because the mines were new or reactivated, were entitled to deduct 1/6 of total net income in determining excess profits tax. In the Act of 1950 this exemption was increased from 1/6 to 1/3 and the metal mines were given the same coverage.

Redemption of Capital

Mining stockholders are denied the right to redeem their investment from dividends as the mines are being depleted. Under the law all corporate

distributions are ordinary taxable income so long as the corporations earnings and profits exceed the amount distributed. As a consequence stockholders generally receive no redemption of the cost of their shares until the mineral deposits of the company are depleted and assets are being liquidated. Thus the shareholders are forced to take long-term capital losses subject to severe restrictions. As a general rule, such long-term losses give little if any tax benefit to the shareholders.

Suggested Improvements

Suggestions for improvement represent unfinished tasks. In addition to these tasks there is the incessant job of maintaining the improvements in tax structure which contribute to a more healthful atmosphere for the mining industry. A proposal for specific agenda follows:

1. Strive to maintain depletion allowances based upon income percentage. In specific cases where the situation is still unfair, strive to procure more equitable rates. This involves the instruction of law makers and the public about facts which justify the allowances—the capital gain point—the facts about the creation of new national wealth and income through discoveries—that the number of discoveries varies with the amount of exploratory work —that percentage depletion allowances tend to increase exploration activities.

2. (a) The removal of the $75,000 limitations on annual exploration deductions and the 4-year limitation.

(b) The procurement from the Bureau of fair and equitable regulations and rulings pertaining to the new exploration and development provisions.

(c) The adoption of regulations or statutes which will convert the adjusted basis as of January 1, 1951 of capitalized exploration and development costs into deferred charges extinguishable at rates commensurate with the exhaustion of mineral deposits served.

3. Procure clearer and more dignified rulings which will permit, without any question, deductions for loss of useful value of capitalized exploratory and development expenditures from income of the years when these capital items lose their utility.

4. Procure amendments, having retroactive application, which will allow deduction for carry-forward and carry-back operating losses without reduction for the excess of percentage depletion over cost depletion of the years to which the deductions are carried.

5. (a) Strive to maintain the improvements and rulings related to the determination of gross income from the property—the inclusion of the several processes—the allocation formula—the transportation inclusion.

(b) Strive for amendments of all percentage depletion rulings which appear unfair.

6. (a) Strive to retain the law relating to exempt income for excess output.

(b) Strive for clarifying and simplifying amendments to the complex, confusing, regulations.

7. Procure satisfactory rulings pertaining to allowances of exemptions for bonus and subsidy payments, particularly as to amounts received under procurement contracts and irredeemable loans.

8. Procure laws which will permit mining stockholders to amortize their investments out of corporate distributions while the minerals are being exploited.

Opinions of Experts

Executives of twenty-one of the large metal mining companies gave the author their opinions on certain specific points pertaining to the subject matter of this paper. The results of this poll are:

Q. Should appreciation realized from conversion of mineral deposits into money through ordinary mining operations be taxed at capital gain rates?

A. Thirteen—yes, on a technical and equitable basis, but no on a practical basis. They believed that the point should be used to justify and defend the merits of percentage depletion allowances.

Six—yes, without qualification.

Two—no, "percentage depletion is adequate."

Q. As a general rule, do percentage depletion allowances fully compensate for taxing such realized appreciation as ordinary income subject to the normal tax, the surtax, and the excess profits tax, instead of at long-term capital gain rates?

A. Nine—no.

Two—yes.

Five—"it would depend on the facts in each case."

Five—"no opinion."

Q. As discoveries of unknown natural deposits create new sources of taxation, and as the allowances of depletion based upon a percentage of income create incentives and provide cash reserves for exploration, is it your opinion that the allowances of percentage depletion as deductions from gross income have, over the years, resulted in a net diminution or a net increase in the flow of revenue into the United States Treasury?

A. Nineteen—"believed there had been a net increase in the revenue to the United States."

Two—"there had been a net decrease."

Q. Have the numerous attacks, starting in 1935 by the Administration, on the percentage depletion provisions deterred exploration activities?

A. Fifteen—yes.

Three—no.

Three—"no comment."

Q. Has the new tax legislation in the Revenue Act of 1951, pertaining to the treatment of exploratory and development expenditures, stimulated and accelerated explorations?

A. Twenty—yes.

One—"it is too early to forecast."

Q. Has this new legislation had any impact upon the policies of your companies which govern appropriations for explorations and/or development?

A. Sixteen—yes.

One—no.

One—"it is too early to make a statement."

Three—"no comment."

Q. Would you regard favorably and sponsor efforts by a committee representing taxpayers (possibly the Tax Committee of American Mining Congress) to procure, through every available legal procedure, regulations which would construe doubtful questions in the new legislation fairly and reasonably? (In this connection, bear in mind the statement of President Truman in his address to Congress on the State of the Union on January 9, 1952, in which he stated that: "We are going to encourage exploration for new mineral deposits.")

A. Twenty—yes, fourteen specifically commending the Tax Committee of American Mining Congress. One—"no comment."

Books for Engineers

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

Metallurgical Engineering Volume I, by Reinhardt Schuhmann, Jr. Addison-Wesley Press. $7.50, 390 pp., 1952.—This first volume, engineering principles, of a two volume work, is intended by the author to meet a need for reorganization of instruction in extractive metallurgy. Its three main objectives are to organize the subject matter according to basic principles and unit processes; to integrate the underlying science of physical chemistry with metallurgical engineering; and to put greater emphasis on analysis and quantitative solution of engineering problems. Vol. I deals with unit processes, stoichiometry, thermodynamics, fuels, fluid flow, heat transfer, pyrometallurgical systems and refractories. Vol. II will take up the individual unit processes.

Initiation and Growth of Explosion in Liquids and Solids, by F. P. Bowden and A. D. Yoffe. Cambridge University Press. $4.50, 104 pp., 1952.—The book is a study of decomposition by impact and by friction of explosive substances and deals mainly with the work of the authors and their collaborators. Research is still in progress but results thus far indicate that the mechanism of initiation is relatively simple. Findings are thoroughly discussed, and the latter part of the monograph describes the way in which a small nucleus of decomposition can grow into a large scale explosion.

Hydraulics and Its Applications, fifth edition, by A. H. Gibson. Constable & Co. Ltd., London. 35s, 813 pp., 1952.—This textbook has been revised and new material added to provide up-to-date coverage. A new chapter deals with the basic principles of the construction and operation of scale models. Section I covers hydrostatics and the physical properties of water; Section II, hydraulic theory; and Section III, hydraulic machinery.

Chemical Engineering Techniques, by B. E. Lauer and Russel F. Heckman. *Reinhold Publishing Corp.* $6.00, 496 pp., 1952.—The book provides a comprehensive assemblage of industrially proven methods for conducting chemical manufacturing operations. Basic procedure rather than specific equipment is emphasized. The book begins with fundamental operations such as transportation and storage. The main material is presented under four sections: Preparation of material for reaction; production and distribution of energy; conditions affecting the chemical reaction; and the separation and purification of materials. Sale, shipment, storage, or other use is also dealt with in the final section.

Management Controls in Industrial Research Organizations, by Robert N. Anthony and John S. Day. Harvard University. $6.75, 537 pp., 1952.—Detailed results are presented of a first-hand study of problems of administrative control of scientific research activities in American industry. Selected operating data for various laboratories are included in an appendix.

Electrical Code Diagrams, by B. Z. Segall. McGraw-Hill Book Co., $12.50. 343 pp., 1952.—This is the first volume of a two volume set and covers wiring design and protection and wiring methods and materials. It is intended as a practical aid when working in accordance with the National Electrical Code. Each code rule is illustrated by one or more diagrams. Supplementary text is included where possible. Both practical and theoretical material is presented. The second volume will deal with special occupancies, equipment, and systems.

Handy Tolerance Tables, by Alexander Michael, Sr. Handy Length Book Co., $2.00, 144 pp., 1952.—The book is designed to minimize the figuring that eats into job time. An included localizer chart eliminates the chance for error in reading by blocking out all the desired figures. It may be especially useful to the tool designer and machine operator who must maintain job tolerances. The first section of the book is made up of 14 tables devoted to standard industrial fractions and decimal equivalents to three places with tolerances for ±.01 to ±.25.

Handbook of Chemistry and Physics, *The Chemical Rubber Co.* $7.50 (student price, $5.50), 33rd edition, 2857 pp., 1951 to 1952.—New tables, formulae, and analyses, formulated, computed, and tabulated by the 173 contributors to this standard work have been included in the latest edition. Among the new tables included in the edition are; atomic constants, atomic absorption coeffients for X-rays, emission spectra of the heavy elements—protactinium, neptunium, plutonium, and americium, and a table of powers of two for exponents from 1 to 100. Section one contains mathematical tables; section two, properties and physical constants; section three general chemical tables; section four, physics data; and section five, quantities and units.

Quarrying Stone for Construction Projects, by H. Streefkerk. Van Riemsdyck Books. $5.00, 159 pp., 1952.—The book begins with general information on rock types and a discussion of exploratory work and development. Types of equipment and methods of operation are covered in detail but specialized information on blasting is omitted. Current practice in commercial quarries is described.

Petroleum Geology, by E. N. Tiratsoo. McGraw-Hill Book Co. $7.50, 449 pp., 1952.—This text covers both academic questions and practical problems of discovering and exploiting oil deposits. The world's oil fields are reviewed, particularly with reference to their structural and stratigraphic history. The latest techniques of exploitation are described, modern surface methods of exploration are discussed and there is a special chapter on drilling fluids.

Soluble Silicates Volume I, by James G. Vail. Reinhold Publishing Corp. $9.00, 357 pp., 1952.—This first of two volumes deals primarily with the more theoretical work on the properties of the soluble silicates. The second volume will be devoted to industrial application of soluble silicates.

Lead in Modern Industries, *Lead Industries Assn.* $1.50, 230 pp., 1952.—The book has been prepared for the purpose of gathering into one volume a large amount of general and technical information for use of metallurgists, chemists, engineers, and executives. Through the use of the assembled information, it is hoped that they will be able to determine whether lead can aid them in their specific projects. No claim for completeness is made. The book begins with a brief history of lead, and continues with an account of the metals travel from mine to finished product. It treats forms and fabricating processes, storage batteries, cables, corrosion resistance, welding, and other topics.

Chemical Engineering Operations, by Frank Rumford. *Chemical Publishing Co. Inc.* $7.50, 376 pp., 1952.—The book's purpose is to introduce processes peculiar to chemical plant operation. The author's approach is to survey a plant which has been developed using the method under discussion. He then gives an analysis of the underlying principles without attempting exhaustive treatment. Where necessary, examples are given to illustrate plant design from these principles. Chapters are devoted to chemical works pumping, distillations, extraction, drying, mixing, filtration, size reduction, and other facets.

Microstructures in Iron Ore Pellets

by Strathmore R. B. Cooke and Thomas E. Ban

The paper discusses the mineralogy, structure, and strength of magnetite pellets fired in air and in a neutral atmosphere at various temperatures. The information obtained from this investigation is used to explain the origin of the structures occurring in pellets produced by pilot-plant operation.

MUCH experimental work has been performed on the mechanism of agglomeration and pelletizing of moist ore and concentrates and on the process of firing agglomerates, but except for the work of Tigerschiöld and Ilmoni[1] much of the published information is empirical, and the literature contains no information concerning the structure of fired pellets. Results of a preliminary study of the microstructures occurring in pellets produced both in the laboratory and on a pilot-plant scale are presented in this paper. Most of the results were obtained from pellets prepared from Mesabi magnetic taconite, but through the courtesy of the staffs of the Minnesota Mines Experiment Station and the Bethlehem Steel Co., fired pellets produced from limonite and hematite ores and from Lebanon magnetite have also been examined. The work has been restricted almost entirely to pellets produced without additives, the exceptions being those in which coal was added to the filter cake before rolling.

Schwartz has described the microstructures occurring in ore sinters produced under reducing conditions.[2] Many of the structures and constituents he describes are common to fired pellets, but the constituents normally occur in markedly different proportions. His statement that "in all satisfactory sinter the amount of hematite is small, perhaps not over 2 or 3 per cent" is inapplicable to pellets produced by pilot plant operation or by any process in which air is not deliberately excluded. Pilot plant pellets contain over 90 pct by weight of hematite, and those produced in the smaller furnaces of the Minnesota Mines Experiment Station contain more than 80 pct of that constituent.

Preparation of Specimens

It is difficult to examine unfired pellets microscopically because of their insufficient coherency to withstand sectioning and polishing. Fired pellets sometimes lack coherency and frequently contain holes which are troublesome during polishing. Impregnation of both air-dried and fired pellets is readily accomplished by the use of methyl methacrylate monomer.[3] The dry pellets are covered in a test tube with excess monomer, to which a small amount of accelerator has been added. When the test tube is placed in a cooling bath and evacuated with an aspirator, most of the entrapped air is removed, and disruption of weak pellets by boiling of the monomer is reduced to a minimum. Polymerization is then effected by immersion of the test tube in a water-bath at about 45°C, this temperature being held for 4 or 5 days. The test tube may then be broken away from the solidified mass and the pellets sectioned by a carborundum wheel or a diamond saw. Where time is important, impregnation is carried out at a higher temperature by Meyer's method.[4]

Air-dried pellets must be impregnated with especial care; otherwise swelling occurs and the solids are displaced. It is more important to be certain that the monomer forms a reasonably complete network throughout the specimen than that all entrained air is removed, and for this reason the vacuum technique may be omitted.

Impregnated agglomerated products and pellets are extremely coherent, section well, and may be polished either by hand or by machine. Hand polishing is not satisfactory for the examination of conditions existing at the boundaries of grains, and for this machine polishing must be used. The Graton-Vanderwilt machine gives excellent results, provided that pressure on the specimen is considerably in excess of that recommended by the manufacturer.

Mineralogy of Fired Pellets

Only two opaque iron minerals, magnetite and hematite, have been recognized in pellets. The complete identity of these artificial minerals with their corresponding natural counterparts is shown by their mineralographic characteristics and their X-ray structures. Wüstite has not been found.

Slag is the most common transparent constituent of fired pellets. Occasionally conditions have been appropriate for the crystallization of fayalite from the slag, and at other times another crystalline sil-

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Discussion on this paper, TP 3428B, may be sent (2 copies) to AIME before December 31, 1952. Manuscript, Feb. 20, 1951. Revised Sept. 2, 1952. St. Louis Meeting, February 1951.

The Transactions papers appearing in Mining Engineering during 1952 will subsequently appear in AIME Transactions Vol. 193, and may be permanently referenced to that volume.

icate, unidentified, has been seen. Both Schwarz' eutectic structure between silicate and magnetite and his crystallographic intergrowth between these constituents may occur.

Structure of Air-Dried Pellets

Pellets were rolled in a drum 2 ft long by 2 ft in diam. Sectioning of green, or wet, pellets for microscopic examination is unsatisfactory; therefore they were dried at room temperature. Re-arrangement of the solids during drying seems to be of negligible importance, for the measured shrinkage during this process is less than 0.001 in. for pellets which are 0.75 in. in diam.

Fig. 1 shows part of an air-dried pellet which was impregnated as described above, sectioned and polished. It was prepared from an eastern Mesabi taconite concentrate, the size analysis and composition of which are given in Tables I and II. Most of the experimental results reported in this paper were obtained on pellets prepared from this material.

A pellet with 25 pct porosity, made from a concentrate containing 91 pct magnetite (65.9 pct total Fe) and 9 pct quartz will contain by volume 62.4 pct of magnetite and 11.6 pct of quartz. Because of this, the magnetite content appears disproportionately low in polished sections when compared with the high iron content of the concentrate. Fig. 1 shows the individual fragmented particles of magnetite and gangue, together with some interlocked particles of gangue and magnetite. Fig. 2, taken with the highest available resolving power, shows the irregular and fragmented nature of the magnetite. Particle size ranges down to 0.25 micron, the limit of resolution of the objective used. Because the objective was an oil immersion lens, both the plastic and the silicate gangue appear dark.

Structure of Magnetite Pellets Fired in a Neutral Atmosphere

Oxidation of magnetite occurs at low temperatures. Under the requisite conditions it will slowly oxidize to hematite at temperatures not materially different from those prevailing at the surface of the earth; in the laboratory marked oxidation occurs in half an hour at 200°C. To eliminate the variable of oxidation, pellets were heated to various temperatures in an electrically-heated tube furnace in an atmosphere of nitrogen or helium from which all traces of oxygen were removed. For this purpose the commercial gas was bubbled through absorption bulbs containing a solution prepared by reducing acidified potassium dichromate with zinc. The resulting blue chromous salt has great affinity for oxygen, changing color immediately when its absorptive powers are exhausted. The success of this treatment became obvious upon comparison of pellets heated in commercial nitrogen or helium with others heated in the de-oxygenated gases. The first-mentioned pellets contained relatively large quantities of hematite, the second, none, except that which could be attributed to the residual air in the pores of the pellets.

The pellets were prepared from concentrates of the same composition and size distribution given in Tables I and II. Air-dried pellets were heated in an inert atmosphere at temperatures ranging from 100° to 1200°C by direct introduction into a fur-

Table I. Size Analysis of Concentrates

Size		Wt, Pct
−28	+48 mesh	0.36
−48	+65 mesh	0.27
−65	+100 mesh	0.50
−100	+150 mesh	1.35
−150	+200 mesh	4.43
−200	+270 mesh	8.30
−270	+34 microns	9.88
−34	+24 microns	10.80
−24	+17 microns	15.31
−17	+12 microns	13.00
−12	+8.5 microns	11.38
−8.5	+6 microns	8.74
−6		15.78
	Total	100.00

Table II. Chemical Analysis of Concentrates

Constituent	Wt, Pct
Total Fe	63.25
Fe^{++}	19.00
SiO_2	8.84
Al_2O_3	1.45
CaO	0.08
MgO	0.57
TiO_2	0.08
Mn	0.12
S	0.01
P	0.025

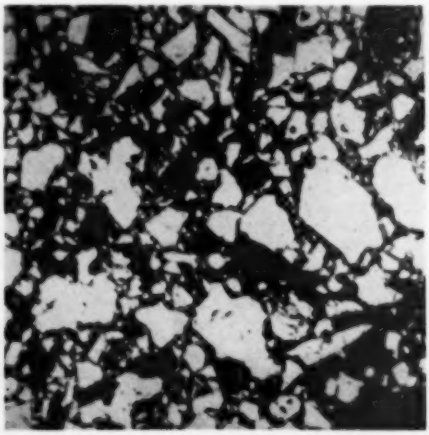

Fig. 1—Section of pellet rolled from eastern Mesabi taconite concentrate. Magnetite is white, quartz and siliceous gangue minerals are grey, and pores are filled with plastic. X250.

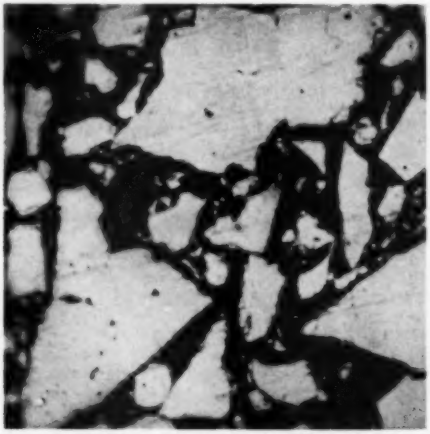

Fig. 2—Section of pellet shown in Fig. 1. X1,200, oil immersion.

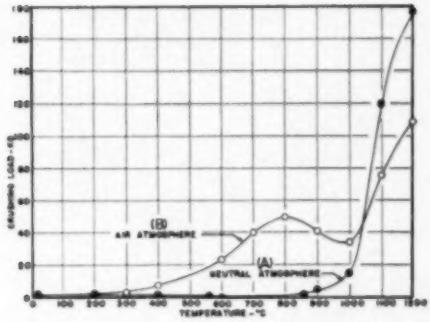

Fig. 3—Strength-temperature curves, 30 min.

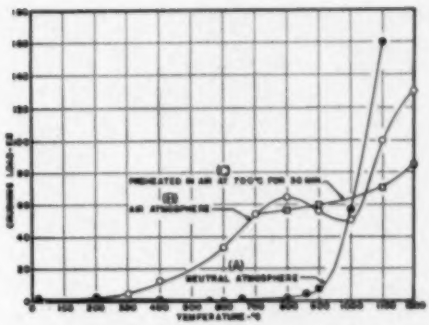

Fig. 4—Strength-temperature curves, 120 min.

nace already at temperature. Two periods of heating were employed for each temperature, 30 and 120 min, respectively. In addition to those pellets selected for microscopic examination, 10 or more pellets were treated at each temperature and their strengths determined in an Amsler testing machine. The arithmetic mean strengths are given in Figs. 3 and 4, curves A.

Air-dried pellets had a strength of 1.2 kg. A surprising feature, shown by both the 30 and 120 min firing tests, is the drop in strength between 200° and 660°C. This is accounted for by removal of the last of the adsorbed film of water responsible for the dry strengths of the pellets.

At 660°C the 120-min series has returned to the original dry strength, and the 30-min series shows a similar response. Between 800° and 900°C both series start gaining markedly in strength, and as shown by curves A in Figs. 3 and 4, the strength increases exponentially with temperature. For comparison, the strength of the best pilot-plant fired pellets is of the order of 200 or 250 kg.

Specimens fired in an inert atmosphere for 120 min at temperature intervals of 100° from 500° through 1200°C were impregnated and polished. The sections were examined microscopically, both at low powers for changes in general appearance and at high resolution to determine the existence of small changes, especially at the grain boundaries. Specimens fired below 800° show only a slight conversion of magnetite to hematite, due to entrapped air, with no structural changes except for a cracking of the hydrated silicates. On the other hand, all pellets fired up to 660°C disintegrated with considerable rapidity when immersed in water, and those fired at higher temperature remained intact indefinitely under the same conditions. The writers regard this as indicating that weak bonding commenced at this low temperature.

Examined microscopically, pellets fired at 800°C show no obvious change compared with the air-dried pellets. At high resolving powers occasional thin blades of hematite can be seen in the magnetite grains, its distribution being uniform throughout the pellet. Since much less hematite occurs, only sporadically, in the untreated pellets, it is assumed that this hematite originates from oxidation by the residual air in the pores, before the nitrogen has had time to sweep it out. Assuming that a ¾-in. pellet contains 25 pct porosity, and that all the contained oxygen is available for conversion of magnetite to hematite, then approximately 7 mg of hematite will be produced. This is of the order of magnitude observed.

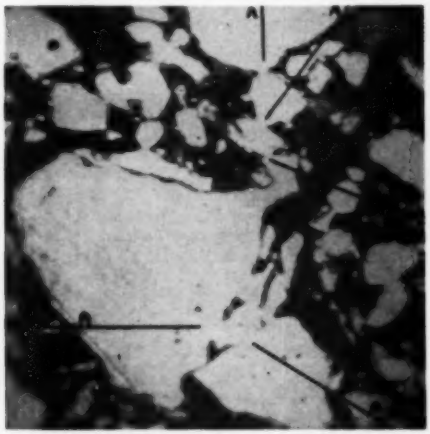

Fig. 5—Bridging of magnetite grains, A, at 900°C. X1000, oil immersion.

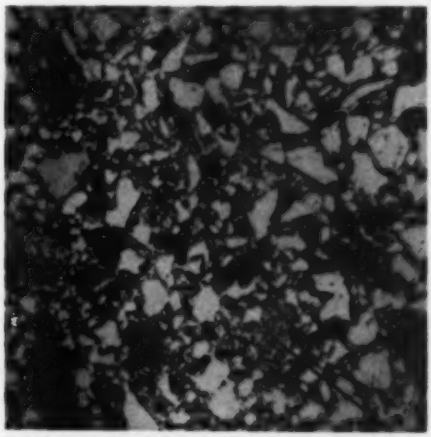

Fig. 6—Section of pellet fired at 1000°C, showing agglomeration of fine magnetite particles, A, and bridging of coarser magnetite. X250.

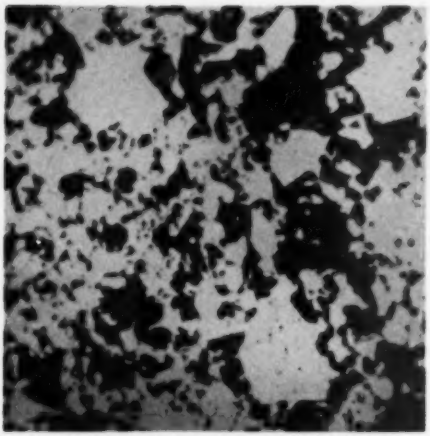

Fig. 7—Section of an iron ore pellet, showing continuous lattice of magnetite at 1100°C. X250.

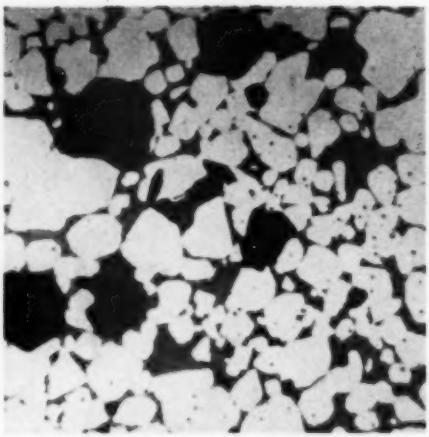

Fig. 8—Magnetite grains and continuous slag network at 1200°C. Compare this with Fig. 7 for changes in pore size. X250.

The 900°C pellets show slight but important differences compared with the 800°C pellets, but these are observable only at high powers. First, the hematite has entirely disappeared, presumably having entered into solid solution in the magnetite. Second, there is a distinct tendency for grains of magnetite which are in contact to join together and form short bridges, as shown in Fig. 5. Thirdly, there is a blunting of the cusps of the original magnetite grains. On the other hand, the distribution of the micron-size particles of magnetite appears the same as in the previous specimen, and there is no apparent change in the gangue.

At 1000°C there are marked changes. First, nearly all the smallest magnetite grains have agglomerated into relatively large sponge-like masses, shown in Fig. 6. Secondly, the bridging between adjoining magnetite particles has become very pronounced, an extension of the structure shown in Fig. 5. The larger gangue particles show no change except for blunting of the cusps, but smaller particles have begun to lose their identity and tend to yield to the forces causing the magnetite to grow.

At 1100°C the original microstructure of the pellets is almost unrecognisable, for in large measure the magnetite has recrystallized. Only the very largest magnetite and gangue particles preserve something of their former identity, and there is a continuous structure of the former from one edge of the pellets to the other. The spongelike masses of magnetite occurring in the pellets fired at 1000°C have become massive, with the internal structure broken only by fine lines of slag. The gangue has been largely converted to glassy slag, forming a discontinuous network between the magnetite grains and acting as a bond between adjacent particles. Fig. 7 shows a section of one of these pellets. It should be noted that attainment of this structure is coincident with a very marked increase in strength. There must be a very considerable mobility of both magnetite and slag at 1100°C, for the many fine pores have coalesced into a much smaller number of large pores.

Fig. 8 is a photomicrograph of a pellet fired at 1200°C. For all practical purposes the magnetite is monodisperse, the grains being surrounded by slag. The pores have been reduced greatly in number, and are now of a size comparable with the largest magnetite grains. Curiously, there is no bridge structure of the magnetite as in the 1100°C pellets and the slag has now assumed continuity of structure across the specimen.

From the preceding observations two inferences appear safe. 1—In a neutral atmosphere, magnetite grains will bond, when and where they touch, at least at 900°C, and possibly at still lower temperatures. Under these conditions there is no visible change in the gangue constituents except dehydration. 2—At somewhat higher temperatures the gangue plays an important part in the growth of the magnetite. At about 1000°C grain growth commences at the expense of the fine magnetite particles which are spacially separated. It seems improbable that such growth can occur unless there is actual transfer of magnetite through another medium, in this case the gangue or the slag which is forming from the gangue. In fact, it is probably no coincidence that marked grain growth of magnetite and increase in pellet strength are contemporaneous with the formation of conspicuous quantities of slag. It is noteworthy, however, that observed cementation of osculating particles of magnetite occurs some 700°C below the melting point of magnetite, suggesting high mobility in the solid state.

The gangue constituents apparently possess considerable mobility at 1000°C, possibly analogous to the softening of a siliceous slag, but seemingly more than 100° too low for melting to occur. Taking the probable slag-forming constituents of Table II, neglecting the small amount of TiO_2, and recalculating to 100 pct will give the results shown in Table III.

Table III. Slag-Forming Constituents in Taconite Concentrates

Constituent	Wt. Pct
SiO_2	70.8
Al_2O_3	11.6
CaO	0.6
MgO	4.6
MnO	12.4
	100

In slag formation, whatever amount of FeO is necessary for equilibrium will come from the complex iron silicates in the gangue and from the magnetite. From the available equilibrium diagrams, the melting point of a slag containing the listed oxides and FeO cannot be much below 1150°C.

Structure of Magnetite Pellets Fired in Air

For an investigation of the effect of oxidation on strength, pellets similar to those fired in inert atmospheres were heated at temperature for 30 and 120 min. Ten pellets were used to determine strength at each temperature for the two series, and the results are plotted as curves B in Figs. 3 and 4.

Strength increases with temperature, reaching a first maximum at 800°C. Thereafter it decreases to a minimum at about 1000°C, and then climbs to a second and undetermined maximum at higher temperatures. The minimum above 800°C was unexpected but is confirmed by similar tests made elsewhere on other magnetite concentrates. Pellets fired according to the schedule given were impregnated, sectioned, and semi-polished on 3/0 metallographic paper, but were not given a final polish because of their striking appearance at this stage. Pellets fired for 30 min at 400° and 600°C were red throughout and macroscopically were uniform from edge to edge. Those fired at 700°C contained a relatively small dark-grey core surrounded by a red shell. The 800°, 900° and 1000°C specimens had progressively larger cores and thinner shells. The cores of the 1100°C pellets approximated in size those of the 900°C specimens, those of the 1200°C pellets were similar in size to those of the 800°C pellets, and both the 1100° and 1200°C pellets contained marked contraction cracks separating cores from shells. All the 30-min pellets were magnetic, the 700° to 1200°C specimens especially so. The structures reported above are shown diagrammatically in the upper part of Fig. 9.

Of the pellets fired for 120 min, those from 400° to 1000°C were uniformly red. The 400°C pellets were magnetic, the 600° and 700°C pellets weakly so, the 800° to 1000°C specimens were non-magnetic, and the 1100° and 1200°C pellets were strongly magnetic. Those fired at the two highest temperatures possessed cores similar to, but smaller than, those of their 30-min counterparts, and the concentric crack between core and shell was well developed, see Fig. 9.

Examination showed that the dark-colored cores consist of unaltered or of very slightly oxidized magnetite, and that the red shells consist of slightly oxidized to completely oxidized coarse magnetite grains in a network of very fine hematite, see Fig. 10. This last-mentioned structure originates in the

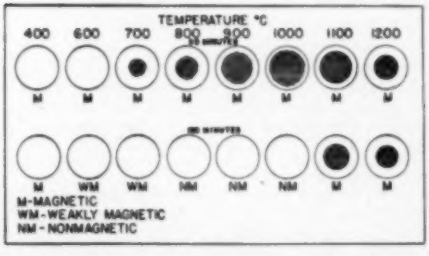

Fig. 9—Macroscopic structures of magnetite pellets fired in air.

complete oxidation of the interstitial slime magnetite, a process occurring rapidly at relatively low temperatures. Internal reflection within this fine hematite gives a predominating red color to this part of the pellet, even though magnetite may be the more abundant mineral. For example, the 400° and 600°C pellets fired for 30 min are magnetic, even though megascopically they are red throughout. The cores of pellets fired at 700°C consist of coarse magnetite grains surrounded by a network of magnetite slimes in which there are only occasional areas of oxidation. The cores of the 800° to 1100°C pellets are similar, but the microscopic structures previously described for pellets fired in nitrogen are becoming evident. At 1100° and 1200°C the magnetite of the cores assumes the structures shown respectively in Figs. 7 and 8.

Nearly all the magnetite in the 120-min specimens fired at 600° and 700°C has been oxidized, and there is no magnetite in those specimens fired at 800°, 900°, and 1000°C.

Although the rate of oxidation should increase with increasing temperature, in the case of the 30-min pellets the amount of magnetite converted to hematite actually decreases with increasing temperature, up to 1000°C. The authors regard this as due to a decrease in permeability. For example, at 400° to 600°C the pellets possess so loose a structure that oxygen readily penetrates the interstitial slimes and oxidizes the fine magnetite to the centers of the pellets. It is presumed that at higher temperatures the freshly-formed hematite is more mobile and bridges and seals the pores so that gaseous penetration is substantially reduced. Sealing is incomplete, however, because total oxidation occurs in 120 min at temperatures of from 800° to 1100°C, inclusive.

The formation of hard cores in both series of tests at 1100° and 1200°C is due to the fact that lowered permeability permits early slag formation and recrystallization of magnetite at these temperatures, just as in magnetite pellets fired in nitrogen. Once slag has formed and the grain size has coarsened, oxidation is necessarily slower.

Concentric shrinkage cracks at temperatures above 1100°C are due to condensation of the core. At these temperatures the slag has softened and pores have enlarged but have become less numerous; eventually they coalesce by surface tension, in the space between the rigid shell and the core. Significantly, no concentric cracks occur in pellets fired in neutral atmospheres, but a decrease in volume of 15 pct or more occurs in those fired at 1100°C or above.

Structure of Pilot Plant Pellets

The characteristics of laboratory-fired pellets have been discussed at considerable length, since they provide valuable information concerning the behavior of pellets fired in pilot plant furnaces. However, laboratory conditions are not identical with those of practice. In large-scale firing, wet pellets are loaded gently onto the charge, which in a matter of inches below the loading level passes through the maximum temperature in the furnace. Expulsion of water is extremely rapid, the pellets reach temperature very rapidly, and are soaked for a long period, measured in hours, in a heat-exchanging, oxidizing atmosphere. In practice, furnace temperatures are high, for heat liberated by the oxidation of the magnetite to hematite is available. Furthermore, coal is frequently added to the concentrates before rolling,

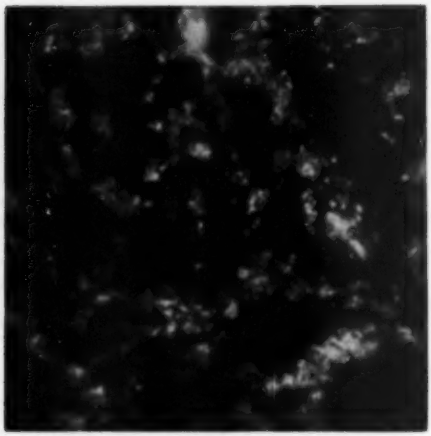

Fig. 10—Network of oxidized slime magnetite, white, surrounding larger grains of superficially oxidized magnetite, black. X1000, oil immersion, crossed nicols.

and this contributes additional heat as well as carbon monoxide, both of which may play a determining role in the mineralogy of the pellets.

Magnetite pellets with or without coal, fired in the Minnesota Mines Experiment Station shaft furnaces, possess structures almost identical with out pellets fired for 120 min at 1200°C in air. The shells contain hematite grains pseudomorphic after the original larger-fragmented magnetite particles, together with the slime network of hematite. The structure of the cores is identical with that shown in Fig. 8, even to the order of size of the recrystallized magnetite. In some cases the core magnetite is retained as such; in others it has been oxidized subsequently to hematite. With a larger amount of gangue present than in the concentrate used by the writers, the magnetite often attains singular perfection of crystal shape.

Pellets made in the large pilot plant furnaces consist almost entirely of hematite. Microscopic study clearly shows that prior to complete oxidation they have a binary structure identical with that previously described. The only apparent changes consist in the replacement of the recrystallized magnetite of the cores by hematite, and, in the highest-strength pellets, a coarsening of the grain of the shell. Oxidation of the core occurs without modification of the outlines of the magnetite polyhedra.

Thermal shock may cause radial cracking of the shell. If this occurs at sufficiently high temperature, the recrystallizing magnetite and fluid slag extrudes from the crack. Grate-fired pellets commonly show this effect, which may also occur in furnace products.

It is generally recognized that *red* pellets are weak and friable, but that dark *blue* pellets are strong and show little friability. There is a remarkable correlation between strength and microstructure. Red pellets may have a shell in which there is a great amount of slime hematite, best seen in polarized light, or may have no core, the entire pellet consisting of oxidized slime magnetite bonding more or less oxidized and unrecrystallized magnetite. Blue pellets, on the other hand, contain little or no slime hematite in the shell, and the cores consist of slag-bonded recrystallized magnetite or of slag-bonded hematite formed from the oxidation of that magnetite. It is certain that either through channelling or some other effect the red pellets have not been heated to a high enough temperature.

The authors' concept of the changes through which pellets proceed during pilot plant firing is as follows:

1—Moisture escapes from the pellet. Either during this operation or shortly following it the interstitial slime magnetite of the shell oxidizes, and by at least 300°C there is an appreciable increase in strength due to cementation and grain growth.

2—As the pellet moves into a hotter zone, oxidation of the slime proceeds inward, and the coarser magnetite of the shell oxidizes.

3—At 800°C the shell formation is almost completed. The shell has considerable hot strength and its permeability to the furnace atmosphere is greatly reduced. The pellet is now subject to appreciable load from overlying burden, but the shell prevents collapse of the dry, powdery magnetite core.

4—Marked changes in the core commence between 800° and 900°C and continue to high temperatures. The magnetite forms a network structure by cementation, and there is softening of the gangue. The hot strength of the core probably increases because the magnetite network is forming in the solid state.

5—At some temperature between 1100° and 1200°C the slag-forming constituents have become plastic and form a matrix in which the magnetite disperses as more or less equiaxed polyhedra. However, if the slag volume is small there is evidence that the magnetite network of item 4, above, is retained. The core presumably has little hot strength, and it is only the oxidized shell which prevents collapse. In many pellets much of the pore space rearranges into a concentric crack between shell and core.

6—In all pellets there has been complete recrystallization of the magnetite in the core prior to the start of oxidation. Oxidation commences at the outside of the core and slowly proceeds inward. In pellets which have been fired in small shaft furnaces oxidation does not necessarily proceed to completion, and the cold pellet may contain a substantial magnetite core. All pilot plant pellets examined, however, contain very little residual magnetite.

Addition of coal to magnetite pellets raises peak temperatures and probably permits formation of a larger core. Excess coal causes fusion and collapse of the pellet.

From the point of view of commercial production of pellets from magnetite concentrates, it is extremely fortunate that a shell containing slime hematite forms in the early stage of firing.

Acknowledgments

The writers wish to express their thanks to Professor E. W. Davis and to Mr. L. S. Taylor of the Minnesota Mines Experiment Station, to the former for the samples of fired pellets used in this investigation and to the latter for preparing the green pellets used in the firing experiments.

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The Preparation of Low-Ash Coal

by Adam L. Wesner and A. C. Richardson

This paper describes the development of a continuous float-and-sink process to produce coal low enough in ash content to be suitable for production of electrodes. The cleaned coal had a combined iron and silicon content of 0.239 pct. The efficiency of recovery was 84 pct.

DURING World War II the demand for electrode carbon was greater than could be met by the supply of petroleum coke available for this use. It was believed that coke made from an extremely low-ash coal might be a suitable substitute for petroleum coke. Requirements for electrode carbon to be used for the production of aluminum are that the silicon plus iron content shall be not more than 0.14 pct of the coke. However, during the period when petroleum coke was in short supply, it appeared that relaxations of the specifications might be allowed to permit 0.4 pct iron plus silicon in the coke. For this work, the specifications were that the coal should contain less than 1 pct of ash, and the combined iron and silicon content of the coal should not exceed 0.28 pct.

Heavy-liquid separations made on a number of coals showed that if certain size fractions of some coals were separated at low specific gravities the resulting float products would meet the required specifications. The objective of this work was to develop a continuous, commercially feasible process for producing low-ash coal. After consideration of various processes, the method chosen was a float-and-sink separation using a solution of calcium chloride as the separating medium.

Eagle Seam coal was used for the experiments. The −10 +35 mesh fraction was found to be the most promising feed for producing the low-ash content coal. A batch of 30 tons of −3/8-in. coal was screened at 10 and 35 mesh; the yield of −10 +35 mesh was 20 pct of the feed. This fraction comprised the feed for the separation tests. In spite of the fact that this portion was screened a second time at 35 mesh, the coal still contained 16 pct that was finer than 35 mesh. Three-fourths of the undersize was in the −35 +48 mesh range.

A representative portion of the −10 +35 mesh fraction was subjected to batch float-and-sink separations to determine the best gravity to be used in the continuous tests to obtain a high recovery of low-ash content coal. The heavy liquid used was a mixture of carbon tetrachloride and benzene which assured complete wetting of all of the particles. As usual in batch separations, adequate time was allowed for each separation so that even the near gravity particles had ample time to separate. The separations were made at increments of 0.01 sp gr from 1.25 to 1.28 and also at 1.32 and 1.59. Each specific-gravity increment was assayed for ash content. From these data it was concluded that 1.27 sp gr was best for the continuous separation of the coal.

The products from the batch separation were composited into float 1.27 and sink 1.27 sp-gr fractions; these were assayed for ash, iron, and silicon. Table I shows the results of the batch separations. The composite float 1.27 fraction contained 66.04 pct of the total weight, and had an analysis of 0.79 pct ash, 0.073 pct iron, and 0.112 pct silicon, or 0.185 pct iron and silicon combined. These data are indicative of the results possible if a perfect separation is made at 1.27 sp gr under static conditions. Inasmuch as the method under investigation is a dynamic system these results could not be obtained. It remained to be determined, however, just how close to the theoretical results the actual continuous separation would be.

Table I. Batch Float-and-Sink Separation of Raw Coal

Product				Cumulative	
Sink	Float	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct
	1.25	4.61	0.82	4.61	0.82
1.25	1.26	27.67	0.73	32.28	0.74
1.26	1.27	33.76	0.83	66.04	0.79
1.27	1.28	11.53	1.44	77.57	0.88
1.28	1.32	10.27	3.96	87.84	1.25
1.32	1.59	9.64	11.90	97.48	2.30
1.59		2.52	60.30	100.00	3.76

Product	Wt, Pct	Ash, Pct	Fe, Pct	Si, Pct
Composite float 1.27	66.04	0.79	0.073	0.112
Composite sink 1.27	33.96	9.44	0.451	2.369
Composite heads	100.00	3.72	0.201	0.878
Feed		3.79	0.193	0.921

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Dry coal, especially fine dry coal, is highly resistant to wetting by water. In calcium chloride solution the resistance to wetting is even greater. As is shown later in this paper, thorough wetting is essential for a good separation. This made is necessary to develop a method which gave thorough wetting of the coal in calcium chloride solution.

Qualitative tests were made on a number of wetting agents in 1.27 sp-gr calcium chloride solution. Most of the wetting agents were eliminated because they formed precipitates immediately upon addition to the calcium chloride solution. Three wetting agents showed some promise, namely Tergitol Penetrant 4, Du Pont Reagent DP 243, and Invadine C. Comparative tests on these three wetting agents showed Tergitol Penetrant 4 to be far superior to the other two in calcium chloride solution. The amount required was 0.75 lb per ton of coal treated.

Having found a suitable wetting agent, the next step was to establish quantitatively how much the separation in calcium chloride solution is improved when a wetting agent is used. Two representative samples of coal were separated in 1.263 sp-gr calcium chloride solutions, one with 0.05 pct Tergitol Penetrant 4 and the other with no wetting agent. In each case a careful batch separation was made between the float product and the sink product. Table II shows the results obtained. With wetting agent, the float product comprised 54.6 pct of the weight and had an ash content of 0.74 pct. Without wetting agent, the float product contained 79.7 pct of the weight and assayed 1.16 pct ash. This experiment shows the necessity for complete wetting in order to prevent a substantial weight of refuse from reporting in the float coal.

Some of the continuous separation tests described later were made by feeding dry coal into the calcium chloride solution, but it was also desired to make some of the tests with coal which had been prewetted in water. Cold water, hot water, and steam were tried, but these did not give complete wetting of the dry coal. Several wetting agents were tried in water; the best wetting agent found was Wetsit. The required amount was about 0.25 lb per ton of coal treated, or about one-third the amount of Tergitol Penetrant 4 required to wet dry coal in calcium chloride solution. As the cost of both

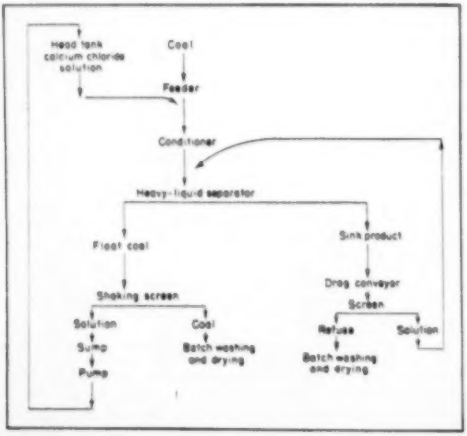

Fig. 1—Flowsheet used for the production of low-ash coal.

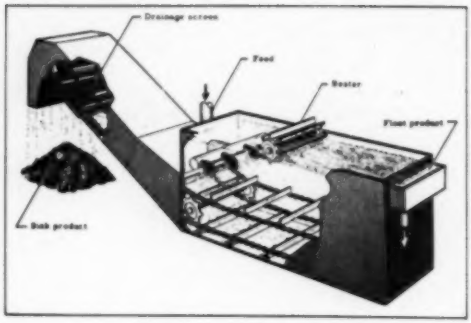

Fig. 2—Heavy-liquid separator.

of the wetting agents was about 25¢ per lb, wetting in water with Wetsit would cost about 12¢ per ton less than wetting with Tergitol Penetrant 4 in calcium chloride solution.

The flowsheet for the continuous-separation experiments is shown in Fig. 1. Calcium chloride solution served not only as the separating medium, but was also circulated through the system to provide a means of transporting the feed coal and the float product.

In the experimental plant, no provision was made for continuous washing and recovery of the residual calcium chloride solution retained on the products. Washing and drying were accomplished in batch operations.

A sketch of the separating vessel is shown in Fig. 2. This is a rectangular settling tank 2 ft wide, 8 ft long, and 3½ ft deep. A drag flight conveyor is provided to transport the sink product across the bottom of the vessel, up the inclined portion of the tank, over a stationary drainage screen, and then to discharge. A slow-speed multiblade paddle beater is provided to agitate the float coal gently to reduce occlusion of refuse in the float coal.

Table II. Batch Float-and-Sink Tests on Coal in Calcium Chloride Solution With and Without Wetting Agent

	Without Wetting Agent		With Wetting Agent	
Product	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct
Float 1.263	79.7	1.16	54.6	0.74
Sink 1.263	20.3	12.95	45.4	6.85
Composite	100.0	3.65	100.0	3.51
Heads		3.79		3.79

In operation, the circulating solution and the coal are fed into a central standpipe of the conditioner, and then discharged from the top of the conditioner by gravity through the side or end of the tank about 1 ft below the liquid level. The feed is distributed evenly across the width of the tank by means of a manifold pipe. Besides the solution entering with the feed, calcium chloride solution is added to the inclined portion of the tank to prevent float coal from being trapped in this section. The calcium chloride solution is maintained at the desired density and volume by additions of either strong liquor or water to the pump sump as needed.

Procedure and Results of Continuous Separations

Preliminary experiments were made to establish the best operating procedure for the continuous separation. It was established that 1—a cleaner

float product was made when the multiblade paddle beater was used, 2—a higher yield of float product was obtained when about one-fifth the circulating solution was added to the inclined portion of the tank, and 3—there was no difference in the results when the two methods of wetting described previously were tried.

In the final, large-scale test the coal was prewetted in water. A six-ton lot of coal was agitated in water containing Wetsit; then the coal was filtered on a pan-type vacuum filter. The moisture content of the filtered coal was 9 pct. The coal was fed to the separating vessel at a rate of 1000 lb of dry coal per hr. The total amount of calcium chloride solution circulated was 2700 lb per hr, 2200 lb with the coal, and 500 lb to the incline. It was necessary to add strong calcium chloride solution to the head tank in sufficient quantity to compensate for the water carried into the system by the wet coal. The specific gravity of the separating solution was maintained between 1.265 and 1.270 throughout the test. The products of the test were washed free of calcium chloride, then dried, weighed, and assayed.

The results showed the yield of float coal to be 66.4 pct of the total feed, the assay was 0.92 pct ash, 0.087 pct iron, and 0.152 pct silicon, or 0.239 pct total iron plus silicon. Using the ash content of 0.92 and interpolating in Table II, the float coal assay would indicate a separation at a gravity of about 1.284 and the yield at this gravity should be 78.7 pct. The yield actually obtained was 66.4 pct; therefore, the efficiency of recovery was 84.4 pct.

The ash content of the float product made in the continuous separation at sp gr 1.265 to 1.270 was slightly higher than that of the float 1.27 product made in the batch separation. The higher ash content is the result of a less perfect operation of a continuous system compared with a batch separation. This difference is to be expected, because in batch operations the near gravity particles have periods of undisturbed quiet in which to find their proper place, while the solution in a continuous separating tank is in constant motion even though precautions are taken to reduce turbulence and agitation to a minimum.

Acknowledgment

The authors wish to thank the Koppers Co., sponsor of the project under which this method was developed, for permission to publish this article.

Cementing in Deep Diamond Drill Holes

by Adrian E. Ross

Diamond drilling through caving formations at depths greater than 2000 ft has long presented serious difficulty, generally solved either by casing or cementing. Casing is permitted only when the resulting smaller hole is acceptable, and cementing is frequently a problem at depth. A new method that injects cement under pressure at the desired location is an aid in diamond drilling deep holes.

DRILLING through caving formations in diamond drill holes at depths greater than 2000 ft has long been a serious problem. These caving formations are normally passed only by casing the hole or by cementing the hole. Casing the hole is possible only when a resulting smaller hole is acceptable. In as much as only a limited number of reductions are economically practical, cementing is frequently considered the only method suitable for solving these problems. Cementing either by pumping a slurry into the hole or by dropping cylinders containing cement has proved satisfactory on holes under 2000 ft. In holes substantially deeper than 2000 ft these methods are not satisfactory, since during the pumping process cement *separates out*, whereas cylinders usually break before the bottom of the hole is reached. A new procedure, *the injector method*, apparently solves these problems.* With this method, cement is lowered in the hole in a cylinder, the cement being

* This cement injector device and procedure were developed by Robert R. Carver, Assistant Manager of Contract Department, Sprague & Henwood, Inc., Lewis W. Fetherman, Factory Engineer, Sprague & Henwood, Inc., and the author.

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Discussion on this paper, TP 3423A, may be sent (2 copies) to AIME before Dec. 31, 1952. Manuscript, Dec. 5, 1951. New York Meeting, February 1952.

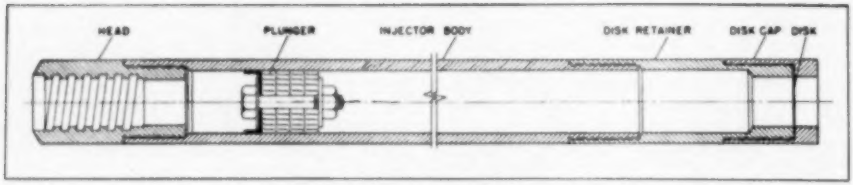

Fig. 1—Design of cement injector.

held in place by a disc in the bottom of the cylinder. When the caving formation is reached, sufficient hydraulic pressure is applied to a piston at the top of the cement cylinder to break the disc, causing the cement to be injected under pressure at the desired location in the hole. In this way, thick, properly mixed, quick-setting cement can be placed anywhere in the hole at the operator's discretion.

The Problem in Drilling

Drilling of deep diamond drill holes becomes an increasingly difficult problem when intermittent soft broken formations or faulted zones are encountered. Such formations or faulted zones cause caving, with the result that the hole is filled each time the core-barrel is withdrawn. The caved section is then re-drilled at high bit cost. Furthermore, it is often difficult or even impossible to advance the hole beyond the caved area. The normal solution to such a problem is either to case the hole, thereby reducing it one size, or to cement it.

Consider first the problem of casing. If only one cave area is encountered, the hole could very readily be reduced one size. For example, assume a hole were being drilled with an NX bit, 2⅛-in. diam core, the largest of the four standard sizes. If the customer would accept a BX core, 1⅝-in. diam, then the hole could be reduced one size to BX and continued. In some instances the customer would accept an AX core, 1⅛-in. diam, which is the next smallest size, and if necessary the hole could be reduced a second time. If he were willing to accept an EX core, ⅞-in. diam, the size following AX, theoretically the hole could be reduced a third time. However, it is considered impractical to drill size EX in a hole 4000 to 5000 ft deep.

It might logically be assumed that a size larger than NX should be used in starting the hole if it is expected that numerous cave areas will be encountered, but at this point economics enter the situation. If holes larger than NX are drilled, the cost per foot is greatly increased. The exact amount varies with the formation and the depths being drilled, but it is safe to say that using the next size larger than NX would render the cost of the hole at least 25 pct greater than the cost of the NX hole. Two sizes larger than NX could very easily increase the cost over NX by 75 pct, and in some cases as much as 100 pct. Such increases in cost naturally limit the use of these larger sizes. Frequently the customer is unable or unwilling to pay the cost of starting larger size drill holes.

In one recent instance, several deep holes were drilled across a number of fault zones and zones of finely crushed mica schist. Each time the core barrel was withdrawn the hole immediately refilled. Two days of continuous washing failed to clean the hole, and casing it was considered economically impractical. Conventional methods of cementing were therefore tried, as described in the following paragraphs.

Conventional Cementing Practices

Conventional methods of cementing diamond drill holes range from pouring cement from a pail to dropping well prepared cardboard cylinders of carefully mixed cement down the drill hole. Two of the most commonly used methods are described below, with comments on their failure at depths greater than 2000 ft.

Pumping the cement mixture through the drill rods is a method now commonly used in deeper holes. One version of this procedure is described in detail by J. D. Cumming.[1] Drill rods with a special coupling attached to the bottom end are lowered into the drill hole. This coupling contains a small opening in the bottom end and other openings around its periphery. The peripheral openings are used to wash the cave from the wall of the hole. When the hole is properly cleaned, cementing is started. For this operation the intake of the pump is transferred from the water source to a container of cement mixed sufficiently thin to permit proper flow through drill rods and drill rod couplings. The required amount of cement is then pumped into the hole and the rods withdrawn and washed before the cement has time to set in the rods.

Attempts to apply this method failed. At depths greater than 2000 ft the cement apparently had a tendency to separate out or *drown*, thus preventing proper hardening. There may also have been a small flow of water in the cave area. This may have washed away the thin cement before it had a chance to set. Then too, at such depths the friction head developed was excessive, and standard high pressure

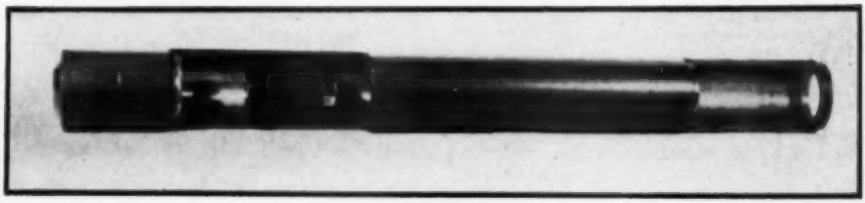

Fig. 2—Cement injector.

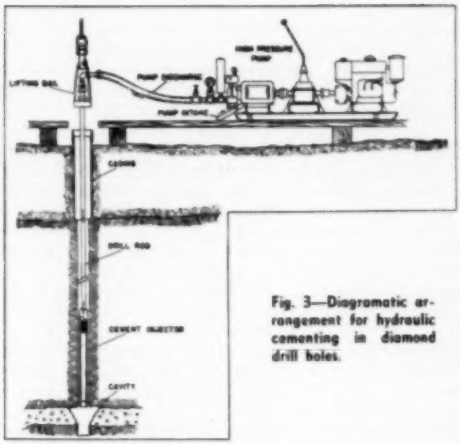

Fig. 3—Diagramatic arrangement for hydraulic cementing in diamond drill holes.

drilling pumps were neither suitable for such cementing work nor capable of developing adequate pressure. Special grout pumps would have been required before this method could have been given a further trial.

In the other procedure normally used for difficult cementing conditions, the cylinder method, cement is lowered to the cave area in a cardboard cartridge or paper cylinder. When this method is used it is customary first to drop a string of casing into the hole to permit the cement containers to reach the bottom without interference. The cartridge or cylinder is then filled with cement and dropped through the casing. If the cylinder does not break upon hitting the bottom of the hole, drill rods are lowered to break it, thus freeing the cement. With this method a fairly heavy mixture of cement may be used, permitting a quick set. It offers a distinct advantage over the pumping method if there is any movement of water in the cave area.

In deep holes, however, difficulty was experienced with this procedure. Apparently the hydrostatic pressure broke the cylinders at these depths, since paper and cardboard were found floating to the surface of the hole before any tamping process had been used.

Cement Injector Development

Trouble was encountered in one of the holes previously mentioned at a depth slightly over 2000 ft. The hole was washed for several days in an unsuccessful effort to clear the cave. Casing it was not practical, as this hole and several others were to go to a depth of between 4000 and 5000 ft. Cementing the cave area was considered the only suitable solution to the problem. The cementing methods described above were attempted without success. It was imperative, if the holes were to be completed at the specified size, that a simple and successful method be developed which would not necessitate costly grouting equipment, special high pressure grout pumps, or the hiring of special crews. Cycle time on deep holes is a major economic factor, so the procedure should require a minimum number of trips into the hole. The overall delay time from cessation of drilling to start of drilling should also be kept to a minimum, as labor costs for drilling crews are very high.

To meet these requirements, the cement injector shown in Fig. 1 was developed by Sprague and Henwood, Inc. It will be noted that most of the parts are made from equipment normally available at a drill or in a nearby small shop. The head is a standard AX-casing to B-rod substitute. The plunger is made from pump buckets, together with a bolt, nut, and washer. The injector body is made from flush-jointed casing. The disc retainer is made from solid stock or from tubing with an inside diameter somewhat smaller than the casing so that an internal shoulder will be provided to limit the travel of the plunger. The disc cap is made from material similar to the disc retainer and the disc is made from thin sheet brass or copper. The thickness of the disc may be varied according to the breaking pressure desired. For example, one disc made from .005 standard brass shim stock broke with water pressure on a laboratory test at approximately 350 lb pressure. This same disc broke at 500 to 600 lb pressure when it was used in the hole and the injector was charged with cement.

The injector body may be any required length. Normally Sprague & Henwood manufactures the injector body in 10-ft lengths, coupling 50 to 100 ft of body, depending upon the amount of cement required to fill any given cave area. Fig. 2 is a photograph showing the key parts of the cement injector. Two cut-away portions show the plunger and the disc. The flush-jointed casing is used so that there may be a smooth wall from top to bottom of the casing to permit the piston to move without interference.

Fig. 3 shows a diagramatic arrangement for hydraulic cementing in diamond drill holes. The high-pressure water pump normally used on a deep drill hole is suitable for this purpose. The pump intake is attached to a source of water. The pump discharge is attached to the drill rod, which in turn is held in place by the usual lifting bail. The bottom end of the drill rod assembly is attached to the head of the cement injector and the bottom of the cement injector is placed opposite the cave area to be cemented. Arrows show the flow of water through the pump intake and the pump, and out the pump discharge down the drill rod to the cement injector.

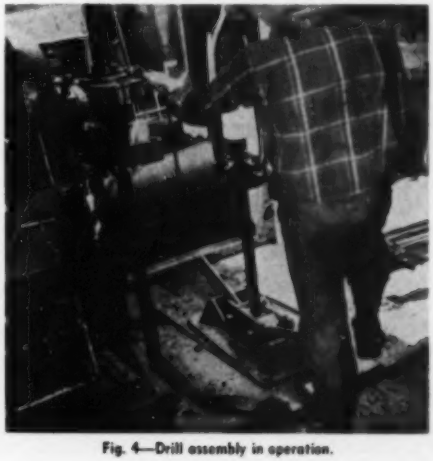

Fig. 4—Drill assembly in operation.

Fig. 5—Drill rig.

Fig. 4 shows a close-up of the drill operator lowering the assembly into the drill hole. The hoisting bail is shown under the operator's arm, and the foot clamp is shown in the bottom of the picture. Fig. 5 is a view of the drill shanty and derrick showing the general arrangement used in the operation described above.

The grout injector method of cementing has been used successfully many times during the past 10 months, usually without any supervisor on the job at the time of cementing. This method has proved successful for cementing cave areas in holes as deep as 4500 ft. It has failed during this development period only a few times, and each time the failure was traced to the operator, who had usually neglected to fill the drill rods with water. Failure to fill the drill rods resulted in excessive pressure under the disc of the cement injector, causing the disc to break prematurely. No other difficulty was encountered.

Cement Injector Practice

When the cement injector is used, the amount of cement required is first determined. In general it has been found that cement in 100 ft of AX casing will fill approximately 65 ft of a BX hole. These figures vary according to the extent of cave area encountered. After the amount of cement required is determined, the type of cement and the mixture are given careful consideration. Portland Cement, Hi-Early Portland Cement, Lumnite Cement, and mixtures of these various types have been tried, as well as admixtures including calcium chloride, plaster of paris, and several chemicals. Lumnite Cement is probably the most satisfactory. Usually the heaviest possible mixture of cement and water is used, with no gravel or sand. The exact mixture depends on the temperature of the day, the temperature of the water, and the time required to lower the cement injector into the hole. Tests are made with the cement mixture to avoid premature hardening of the cement. The operator must have time to insert the rods in the hole, apply pressure, and inject this cement into the cave area. At a shallow depth where little time is required to reach the bottom of the hole, 4 gal of water are used to a bag of Lumnite cement. Under normal conditions, this mixture will provide for 5 hr before the cement takes its initial set. Where the holes are deeper and more time is required to reach the bottom, thinner mixes are used. However, any consistency which is thinner than 7½ gal of water to a bag of cement should be avoided.

When the cement is ready, the disc, disc cap, disc retainer, and first 10-ft length of injector body are assembled and lowered into the hole. It will be noted that the disc retainer has a shoulder which prevents the plunger from passing that point. When this first length is lowered into the hole it is filled with the cement mixture described above. A second length of injector body is then attached, lowered in the hole and then filled with cement. The process is repeated until the correct number of injector bodies are connected to provide the desired quantity of cement. After the last length of injector body is added and filled with cement the piston is placed above the cement and the head is attached to the top end of the injector body.

Drill rods are then attached to the injector head and the tools lowered into the hole. Each section of drill rod is filled with water after it is attached to the string of tools and lowered to the drill platform. When a 40 or 50 ft length of drill rod is used as shown in Fig. 5 some difficulty is experienced in preventing air pockets, but this trouble is held to a minimum if care is exercised in filling each section with water so that pressures on the top and bottom of the disc are equalized. When the cement injector has reached the depth required, the string of tools is held by the foot clamp, a standard waterswivel is connected to the tools, and the tools are then held with the hoisting bail or other suitable means.

The standard high-pressure water pump used on the job is then connected with the string of tools in the hole and the water pressure increased until the disc is broken. It will be noted in Fig. 3 that a water pressure gage is shown on the left end of the high pressure pump. This gage is watched at all times during the operation because it is the key to the satisfactory operation of the cement injector. When the disc breaks, the tools are then pulled back slowly as the cement is being ejected from the injector body. The pressure gage on the pump shows a decrease when the disc is broken, and the pressure falls while the cement is being expelled. When the piston is pushed to the bottom of the casing and shoulders on the inward projection of the disc retainer, the pressure immediately builds up. If the action is too rapid, the relief valve which has been set at approximately 800 lb opens, signalling the operator to turn off the pump. The tools are then withdrawn from the hole and washed thoroughly. The cement is allowed to set approximately 16 hr.

Conclusion

The cement injector described herein has worked satisfactorily to depths as great as 4500 ft. There is no indication as yet that it has been employed at its maximum usable depth. In fact, it is believed that the injector should work satisfactorily at almost any depth, provided a method may be developed for placing it in the hole before the cement takes a premature set in the injector. Field experience alone will determine the ultimate limits of this deep-hole cementing device.

Reference

[1] J. D. Cumming: *Diamond Drill Handbook*, pp. 151-152. Havelock, Ontario, 1951.

Discussion*

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* TP 3421

† For further discussion of this paper see *Transactions* AIME (November 1951) **190**, 991-992.

Pipeline Transportation of Phosphate

by I. S. Tillotson, R. B. Burt, and J. A. Barr

DISCUSSION

Howard Howie *(Knoxville, Tenn.)*—The authors are to be congratulated on the presentation of a paper containing so much valuable information on the pipeline transportation of phosphate, as there is very little literature on the subject.

The writer is especially interested in the paper, as he conceived the arrangement of the Akin and Godwin plants and was in charge of the design work and the engineering incident to their construction.

The Akin and other phosphate deposits in the Tennessee phosphate area lie on beds of limestone that are very irregular. The limestone beds, after the phosphate matrix has been removed, are similar in appearance to land severely eroded by the action of water and denuded of top soil. Depressions in the limestone, called cutters, are irregular in depth with vertical or overhanging walls, having the general appearance of cracks in dried clay. They change abruptly in direction, width, and depth, and vary on the Akin tract from 1 to 25 ft in depth and from 1 to 50 ft in width. Pinnacles of limestone commonly occur in the cutters which appear, when exposed, like small clifflike islands in a river. Limestone floats also occur.

The phosphate matrix fills the cutters and covers the uncuttered areas, the thickness of the cover varying continually and sometimes rather abruptly. It occurs generally as stratifications of phosphate rock and clay of varying thickness.

The phosphate rock in the matrix varies in hardness and in percents of silica, lime, iron oxides, and fluorine, and the clay varies in toughness. In some places the deposit consists of narrow strata of rock almost devoid of clay streaks. In other nearby locations the clay will predominate. When it is excavated, the phosphate rock breaks into thin irregular lumps, locally known as plate rock. Limestone lumps are also excavated with the matrix. Akin plate rock is generally much softer than that occurring in other deposits in the area.

Because of the above described physical and chemical variations of the excavated material, the resultant slurry varies in size distribution, specific gravity, and percent of slimes. When the rock is soft or when there is an increase of clay, the slime fraction is greatly increased as it passes through pumps and pipelines, resulting in reduced pipe friction. It is obvious that the longer the pipeline the greater the reduction of coarse fractions into fines, causing a decrease in pipe friction that cannot be accurately evaluated.

The matrix is mined with a dragline that drops it into a hopper with a grid composed of 9-in. parallel bar spacings located above the hammer mill. The matrix on, and passing through the grid, is subject to the action of powerful sprays which wash it down to the hammer mill, together with any limestone lumps that are not removed before passing through the grid. The hammer mill reduces the feed to lumps of plate rock and clay, most of which will pass through the 8-in. pump suction. The mixture discharges into a pool containing the pump suction pipe. Water from a hydraulic nozzle moves the mixture to the pump suction intake. The pump, driven by a variable speed motor, is the same size as the pumps mentioned on p. 279. Provision is made to remove any lumps that lodge in a bend in the suction pipe in a manner similar to that used in the Florida phosphate fields.

The hammer mill and pump units are mounted on wide steel skids so that they can be moved as the mining operation progresses. The discharge from the pump flows through an abrasion-resistant spiral welded steel pipe 8¼ in. actual inside diam, 8 in. nominal diam, for a maximum distance of 2200 ft, which is the limiting pumping distance for one pump. This pipeline, referred to hereafter as pipeline A, discharges into a ball mill without balls, which in turn discharges into a rotary screen attached to it that separates the slurried matrix into 1¼-in. oversize and undersize fractions. The oversize is returned to the mill for further reduction; the undersize is pumped to a Dorrco washer and then flows into a hydroseparator 160 ft in diam. In spite of the size reduction in the hammer mill and the blunging and washing of the slurry in its passage through the pump, pipeline, mill, and washer, the discharge to the hydroseparator frequently contains mud balls almost perfectly spherical. Sometimes the discharge from the 16,000-ft pipeline at Godwin contains mud balls the size of bird shot and smaller. This pipeline will be referred to subsequently as line B. Liquid caustic is added to the slurry at the Akin plant before its passage through the hydroseparator, which decreases the size of particles in the overflow by dispersion. In passing through pumps 1, 2, and 3 and pipeline B, the slime fraction in the underflow is increased by abrasion and blunging and also by continuing dispersive action of the caustic.

The matrix for use in the experimental tests referred to on p. 279 was obtained from three small surface openings on the Akin tract that were made previous to the purchase of the tract by the Authority. Matrix used in the 2 and 4-in. experimental pipeline tests was taken from the three openings and proportioned to obtain a sufficient quantity that would be fairly representative of the average in the Akin deposits. Prospecting samples of matrix had been obtained from drill holes which showed no small variation in physical and chemical properties. Some of the physical variation is evident from the size distribution of solids in samples taken during the tests covering line B flows so thoroughly made by the Authority under the direction of Mr. Burt, see Table V, p. 280.

Hydraulic gradients for a pipe of 8-in. diam were derived from the 2 and 4-in. pipeline tests using the so-called representative matrix as above described, and plotted on the profile of pipeline B.

Gradients of other materials in slurry form passing through pipelines that bore some similiarity to the Akin matrix slurry were also plotted. After a study had been made of the hydraulic gradients plotted on the profile and the varying slurry flow that would probably occur during actual operation, three pumps were ordered, referred to as No. 1, 2, and 3 on p. 279.

No. 1 and 2 pumps were installed and the third kept in reserve should the operation of 1 and 2 pumps prove satisfactory, since installation of the third pump would require an attendant, as well as the laying of 5400 ft of pipe to supply it with seal-water from the Akin plant. Subsequently, it was found desirable to install the third pump to maintain capacity when the slime fraction was low and the coarse fractions were large.

Reference to Fig. 6 will show that the flow in B line goes upgrade in three locations. At the outlet at Godwin, the slurry flows between two 45° bends for an approximate distance of 18 ft to rise above the ground a sufficient height to discharge into a launder feeding the first classifier. This condition requires extra energy, which is taken care of by keeping the hydraulic gradient a sufficient distance above the high points.

Although there are rather heavy upgrades in the line, the choking condition that might occur at the

low points is mitigated by the outlet's being 109 ft lower in elevation than the inlet.

Referring to Table V, p. 280, during test period 15 the heaviest head loss was 3.14 pct, which is the slope of the hydraulic gradient. If the elevation of the pump is 729 ft, see profile, Fig. 6, p. 278, and the equivalent head of pulp is 395 ft, the head elevation of the pulp would be 729 ft plus 395, or 1124 ft. The head loss in 16,000 ft was 160 times 3.14, or 502 ft, and the gradient elevation at Godwin was 1124 ft minus 502, or 622 ft. The outlet at Godwin is at elevation 620, so the pressure at the outlet was 2 ft, or about 1 lb. If the pipeline were on a level grade at elevation 729, the pressure would be zero at 12,580 ft from pump No. 2, which is located at the Akin plant.

When the pumps stopped operating the first time because of electric power failure, the Akin plant supervisor telephoned to inquire when electric service would be resumed and was informed that it would be an hour. He motored down the road roughly paralleling the line to close the automatically controlled dump valves and to view the results of the discharge through them. He found that they had not opened. When pumping was resumed without the feeding of any additional slurry for some time, the settled solids moved out of the line without mishap. At a later date 3 hr elapsed after an unscheduled stoppage of the pumps before pumping was resumed. The dump valves were not opened and no choking of the solids occurred that prevented subsequent normal operation.

The following sentence on p. 279 is quoted from the paper. "In actual practice, however, the automatic operation of the valves was not entirely successful so they are now manually operated, although dumping is not frequently necessary."

Wood stave pipe was selected for the following reasons: 1. Wood stave pipe had been successfully used for many years for moving tailings from nonferrous plants in the West.

One company had used a 12-in. wood stave pipeline for 4 years without a leak, the flow consisting of rather coarse tailings in a 4 to 1 pulp. The wear on the pipe was not known as it was buried. When the pipe line was laid, a piece of iron pipe was installed between the tailings discharge and the intake to the wood stave pipe. The iron pipe did not last a year.

2. The British escaped from Dunkerque on May 29, 1940, and on Jan. 6, 1941, President Roosevelt asked for billions for defense of Britain and Allies. The pipe was requisitioned some time between these dates when steel was in demand and deliveries uncertain. Abrasion resistant steel pipe was ordered for pipeline A and a sufficient amount for a comparative test with wood stave pipe in pipeline B.

3. The comparative merits of abrasion-resistant steel and wood stave pipe in transporting Tennessee phosphate slurry were unknown. The installation of some of both types would present an opportunity to ascertain which was the more desirable.

The wood stave pipe was made of wire-wound creosoted vertical grain Douglas fir strips 1½ in. thick, with a coat of asphalt on the exterior, the manufacturers claiming that pipe made of creosoted staves lasted longer and had a lower friction loss per 100 ft of length for equal rates of flow.

Abrasion occurred mostly in the bottom quarter of the pipe circumference. Laying the pipe on the surface permitted revolving it to present another nearly unworn surface. A dispersant was not used while the wood pipe was in service.

After about 4 months the wood stave pipe was disconnected and examined. The bottom quarter of the circumference was worn away to a depth of ¼ in. Places occurred for a foot or less in length where the wear was much deeper. Leakage occurred first in those places.

B-Minerals Beneficiation

The Effect of Zinc Deleading Operations on Lead-Zinc Selectivity at the Parral and Santa Barbara Mines

by C. L. Boeke and G. G. Gunther

DISCUSSION

I. M. Symonds, *(Cia. Minera de Penoles, Monterrey, N. L., Mexico)*—Messrs. Boeke and Gunther have written most interesting papers describing their zinc deleading operations. Mr. Gunther was kind enough to show me his operation last fall. Aside from the excellent results, I was most impressed by the ease of control. I had expected a difficult operation requiring careful control and supervision, but found one requiring little attention.

The only disappointing fact of the separation is the poor copper recovery in the concentrate of the deleading float. The final zinc concentrate still has about 3 pct chalcopyrite, assuming the copper is chalcopyrite, and adversely affects the zinc grade. If the copper mineral is free, the poor recovery is probably due to the high cyanide concentration. Fortunately, about 75 pct of the copper is recovered before the pulp goes to the zinc circuit.

Mr. Gunther's remarks about proper cyanide to zinc sulphate ratios cannot be overemphasized. Although our experience has been confined mostly to the conventional lead-zinc separation, we have found these ratios most important, and more important in the mill than in the laboratory. The laboratory appears to give optimum results over a wider range of reagent control than the mill, especially in regard to the effect of cyanide.

In handling relatively high grade ores in the mill we usually find that ratios of cyanide to zinc sulphate of 1 to 10 or 20 or higher work best. Where there is a substantial amount of chalcopyrite present it has been noted that cyanide in the range of 0.02 to 0.10 lb per ton of ore is often beneficial in zinc rejection and possibly in copper recovery in the lead concentrate, while larger amounts, say 0.20 to 0.25 lb per ton, will float more zinc and cause a serious rejection of chalcopyrite. On an ore with 15 to 20 pct zinc content we have had to use 0.8 to 1.2 pounds of zinc sulphate per ton of ore with the above cyanide to obtain optimum results.

Some time ago we made a laboratory study following Parral and Santa Barbara practice on an ore which had never a very satisfactory rejection of zinc in the lead concentrate. The results were not encouraging and a mill test was not run. Since then the study has been reopened using in the laboratory fresh zinc concentrate from the mill instead of one made in the laboratory. A reasonably clean lead float from the zinc concentrate was obtained. It has been checked several times and each time the zinc concentrate made in the mill responds and the one made in the labora-

tory does not. Although undoubtedly some reasonable explanations can be given, it is presented here only as an interesting observation that may be of use to others.

L. H. Lange *(The Galigher Co., Salt Lake City)*—I would like to add to this an experience of mine along the same lines, except a little more exaggerated, on a property in southwest Africa treating dumps which were very rich in copper, lead, and zinc. On a certain fraction of these dumps, some 200,000 tons, averaging about 4 pct copper, 6 pct zinc, and 10 to 12 pct lead, selective flotation was practically out of the question. A technique was developed for the treatment of these ores in which a bulk float of the sulphides was made, the cleaned bulk concentrate being treated with dichromate and American Cyanamid reagent 610 for the depression of the lead and copper sulphides. Dichromate alone had but little effect, but the combination of this reagent and 610 produced quite a satisfactory result. In the early work, selective flotation of some of the easier floating sulphides of lead and copper was attempted, but without much success. However, on some of the upper level mine ores this scheme is now being used before the bulk flotation of the sulphides of copper and lead, more difficult to float, and the easy floating zinc. Sufficient soluble copper is present in these dump ores so that no copper sulphate was needed to activate the zinc, the bulk flotation being carried out with soda ash, xanthate, and frother. This procedure does not make so high a grade zinc concentrate as is made at Santa Barbara, but the company had a very good contract for their zinc concentrates in Belgium, getting pay for the copper and lead in the concentrates. This scheme of treatment developed therefore paid off on these difficult oxidized sulphide ores.

Norman Weiss *(American Smelting & Refining Co., Salt Lake City)*—Since I know the Santa Barbara territory well, I am somewhat familiar with the deleading operations in that district. At Santa Barbara we started deleading in 1941 and 1942 when we ran into some oxidized ore from the Coyote mine. We found it difficult to float all the lead without floating zinc also.

At that time the zinc concentrate ran about 53.5 pct zinc and 3 pct lead. This product was not acceptable at the zinc-smelter because of its high lead content. Consequently, after some experimentation we put in a deleading section.

The zinc concentrate was first conditioned with cyanide and zinc sulphate, also a small amount of soda ash. Sodium bicarbonate and sodium hydroxide were also used as the alkaline reagent. After the concentrate had been conditioned a lead froth was made, and the resulting tailing, zinc concentrate, ran only 1.8 to 2 pct lead, which was satisfactory.

Very many lead-zinc mills now successfully separate lead from the zinc concentrate in some way or another, particularly in the Northwest. Some take a zinc flotation middling and refloat this for lead; others have filtered and dried the zinc concentrate before deleading.

However, the contribution that Mr. Gunther and Mr. Boeke have made to this art is very important, since for the first time they have deliberately permitted lead to escape from the lead circuit in order to improve selection there and have then picked it up out of the zinc concentrate. As far as I know, this is the first time that this has been done.

This seems to be a valuable technique for lead-zinc ores containing copper. On the normal lead-zinc ore containing little or no copper, the zinc content of the lead concentrate may be held below 3 or 4 pct without much difficulty. When the ore also contains copper the problem is more difficult, and the operator has the choice of accepting a lead concentrate containing 7 or 8 pct zinc or even more, or else of upgrading the lead concentrate and sacrificing precious metals and copper.

The method used in the Parral district and described by Mr. Gunther and Mr. Boeke, therefore, may be successfully used on other ores of this type.

H. S. Wildman *(A. S. & R. Co., Avalos, Chihuahua, Mexico)*—At the present time at the Santa Eulalia Mill we are treating the leasers' ore from the Santa Barbara district. As these ores are obtained at or near the surface they contain a great deal of the semi-oxidized or slow-floating lead which tends to report in the zinc concentrate.

Since we have not yet been able to install a zinc concentrate deleading unit like that at the Santa Barbara mill, we have resorted to the tabling of the lead tails prior to zinc flotation and have been able to recover a fair portion of this so-called slow floating lead. At the same time we were able to stabilize the lead float, as the previous tendency had been to add a surplus of collector agent to prevent so much lead passing over to the zinc float and at times lowering the grade below the shipping requirements.

C. F. Zeuch *(The Dow Chemical Co., California)*—In all this work, has anything been done in heating the zinc concentrate before attempting to delead it?

G. G. Gunther (author's reply)—No work has been done at Parral or Santa Barbara to investigate the effect and the possible use of heat in the zinc deleading operations, mainly because a cheap source of heat is not available at either plant.

E. H. Crabtree *(The Eagle Picher Co., Okla.)*—First, I might answer Mr. Zeuch's question on the effect of heat on differential separation. We have one operation now which is being conducted on an old slime pond. This slime pond resulted from muddy primary thickener overflows, which had accumulated many years because the thickeners were too small for the tonnage being handled. A couple of years ago we attempted to clean out this slime pond and retreat it. It contained several percent of combined lead and zinc. Since we were unable to make a satisfactory selective separation by conventional means, we assumed because the extremely fine sizes, we finally wound up by making a bulk lead-zinc concentrate which is cleaned and filtered and then conditioned in what we call our hot conditioner. We give this bulk lead-zinc concentrate a retention time at boiling temperature for approximately 1 hr with high concentration of sodium metasilicate. After this conditioning period, the material is passed through a hydroseparator for removal of the excess sodium silicate and the high caustic solution which we have there. The underflow from this hydroseparator then goes into a lead-zinc differential circuit and we are able to make a reasonably good separation. By that I mean that we make about a 65 pct lead concentrate and about a 58 pct zinc concentrate. This is, I think, a solution of a particular problem. We have attempted this hot conditioning with sodium silicate on other ores and it was not successful, but it is particularly successful on extremely slimy material, which, incidentally, has quite a bit of organic material in it resulting from the original ore.

There is another comment I want to make on Mr. Gunther's paper. As we all know, and as Mr. Weiss pointed out, the main virtue of zinc deleading is that we do not have to control our lead circuit so well. We can work more for lead grade and allow some of the lead to go into the zinc concentrate because we have an opportunity to recover it later. In all the lead-zinc operations I have had experience with, I have found that once the lead goes through into the zinc circuit, which is usually a high-lime circuit, the lead becomes very sluggish and floats there slowly. As a result a large portion of the lead floating in the zinc rougher circuit floats down toward the tail cells. At one of our operations in Arizona we have taken advantage of this. We pull off a zinc rougher concentrate which goes to the zinc cleaners and then to filtration, but the zinc scavenger concentrate, which contains the large portion of the lead, is reground in cyanide and then returned back to the head of the lead circuit. I think one virtue of this is that we can grind in quite a high cyanide strength there without fear of depressing chalcopyrite, since any chalcopyrite that is depressed along with the zinc we have an opportunity to recover again in the lead rougher circuit.

Jaw Crusher Capacities, Blake and Single-Toggle or Overhead Eccentric Types

by D. H. Gieskieng

SUPPLEMENT

D. H. Gieskieng (author)—Under the subject of Crushability-Capacity Effect, on page 972, reference is made to Dodge-type crusher data obtained with jaw strokes of 0.04 and 0.02 in. The latter figure was that employed by Hersam[3] and is the basis for much of the material under this subject in Taggart.[4] The jaw stroke of 0.04 in. was used by Mörtsell[5] in a similarly thorough investigation.

[3] E. A. Hersam: Factors Controlling the Capacity of Rock Crushers. *Trans.* AIME (1923) 68, p. 463.
[4] A. F. Taggart: Handbook of Mineral Dressing, Sec. 4, p. 12. John Wiley and Sons, New York, 1945.
[5] Sture Mörtsell, Bergsingeniör. Prof. Vid K Tekn. Hogskolan.

Screened Ore Used for Fine Grinding at Lake Shore Mines

by Bunting S. Crocker

DISCUSSION

Fred C. Bond *(Allis-Chalmers Manufacturing Co., Milwaukee)*—This is a very complete description of the screened ore fine grinding installation at Lake Shore. The only important data which might be desired are the operating screen analyses and power consumption in each stage.

The statement that the energy consumption, or kw-hr per ton, remains constant for all mill diameters is of great interest, but the opinion is widely held that the kw-hr per ton decreases as the mill diameter increases. More definite test work is needed to establish this point.

Screened ore fine grinding has been successful on hard Lake Shore ore and on about equally hard South African Rand ore. However, it is still not determined whether it would be successful on ores of average or less than average hardness. It is possible that with softer ores the consumption of ore pebbles might be exceedingly high, or might even exceed the available supply. Also, the softer feed might result in a rapid buildup of harmful intermediate sizes in the mill circuit. These factors can be determined only by a full scale or long pilot mill test. In any case the Lake Shore staff is to be congratulated for this pioneering work.

Bunting S. Crocker (author's reply)—As requested, the operating screen analyses and power consumption for each of the four stages of pebble milling are given, see Table XII. The first three stages are in series circuit; the last stage is in closed circuit. The primary circuit had rock feed averaging 124 g per piece and the final stages rock feed weighing 63 g. The intermediate stages had mixtures of the two.

Regarding Mr. Bond's statement that "the opinion is widely held that the kw-hr per ton decreases as the mill diameter increases," I would say that I have frequently been told this but have never yet been shown any evidence or seen any evidence published that would prove this statement. It may be true that some of the big mills recently installed have mechanical power savers such as micarta bearings and better lubrication, but as far as the power consumed inside the mill is concerned, we at Lake Shore think that it varies as $D^{2.5}$, and that is all. We published extensive information on this point in our 1939 paper. Recently Mr. Banks at Consolidated published the figures on his 10 ft 8 in. x 12 ft 0 in. rod mill. As reported in this discussion by the writer,[6] we calculated the power necessary to run this mill based on our 7x6-ft rod mill to within 1 pct of the actual power reported by Mr.

Table XII. Pebble Mill Screen Analysis, Feb. 19, 1952

		Primary Stage, P.M. No. 4* to 7 and 8, Total KW, 385		Secondary Stage, P.M. No. 9 and 11, Total KW, 253		Tertiary Stage P.M. No. 14, Total KW, 121		Final Stage, P.M. No. 17 and 18, Total KW, 261	
		Sand, 700 Tons		Sand, 540 Tons		Sand, 370 Tons		Sand, 310 Tons	
	Mesh	Feed	Discharge	Feed	Discharge	Feed	Discharge	Feed	Discharge
	6	0.2							
	8	1.6	0.2	0.4					
	10	6.7	0.4	1.1	0.2	0.3	0.2	0.2	0.1
	14	16.5	1.2	3.0	0.2	0.5	0.1	0.1	0.1
	20	16.0	1.4	3.3	0.2	0.4	0.1	0.3	0.1
	28	12.8	2.0	4.0	0.2	0.4	0.1	0.3	0.1
	35	12.0	3.9	6.9	0.6	1.0	0.2	0.4	0.1
	48	7.8	5.0	7.8	1.2	1.9	0.3	0.8	0.3
	65	6.5	7.1	11.4	3.0	4.8	0.9	2.5	0.4
	100	5.0	9.7	13.4	7.5	12.2	3.3	6.5	1.4
	150	3.8	11.0	14.1	13.3	20.7	10.0	13.3	3.8
	200	1.7	8.5	11.5	9.4	16.9	15.8	20.1	2.9
	270	2.5	7.7	6.9	15.5	18.2	14.7	23.9	17.9
	325	1.7	9.3	8.3	15.1	14.2	17.7	19.1	22.9
	−325	5.2	32.6	8.0	33.6	8.5	36.7	12.5	50.2
Cum + 3									
Cum + 10		8.5	0.6	1.5	0.2	0.3	0.2	0.2	0.1
Cum + 35		65.8	9.1	18.7	1.4	2.6	0.7	1.3	0.5
Cum + 150		88.9	41.9	65.4	26.4	42.2	15.1	24.4	6.1

* No. 4 P.M. is 6x16 ft, all others 6 ft 8 in. x 16 ft.
Rod mill tonnage, 940 tons per day.

[6] Trans. AIME (1939) 134, p. 97.

Banks. Our formula was $D^{0.6}$! We feel that this point is established.

At the time the paper was given no work had been done on any ore except Lake Shore. The Rand ores are not nearly so hard as Lake Shore. However, recently the writer has had an opportunity to test six different ores, two of which were quite soft; one contained shale. In the preliminary tests these softer ores made much better looking pebbles, if this is important, than did the Lake Shore ore. They did not produce large intermediate sizes, and in fact they cleaned up the "chips" better than harder ores. The softer ores made much "rounder" pebbles and as a result pebble grinding could start at coarser stages than at present practiced at Lake Shore, where 8 mesh is the top size in the feed.

* Canadian Institute of Mining and Metallurgy *Bull.* (July 1952).

J. Myers *(Tennessee Copper Co., Copperhill, Tenn.)*—I was a reader on this paper and consequently had a chance to study some of the aspects of the work. There are three questions that I would like to ask. First, do you think that this means of comminution would apply to ores that are softer than the very hard ore of Lake Shore? The substance of this question is that while the rock from the ore might be softer, the work to be done is less. Secondly, you have laid a great deal of stress on getting the correct size of pebbles. Since you prepared this paper, has any further information developed that would be of aid to any of us who wished to make tests in this manner? Thirdly, have you any idea what the upper range of grinding might be? Many of us would not want to grind quite as fine as you are doing.

Norman Weiss *(American Smelting and Refining Co., Salt Lake City)*—Mr. Crocker kindly let me have a copy of his manuscript some months ago, and we immediately started work at our Neptune property, where it costs us about $60.00 a ton to fly balls into the property. Naturally, we have not made any progress commensurate with what Mr. Crocker has done. We have merely made a very small start, but even thus far the data show that we are going to check Mr. Crocker very closely. The ore at the Neptune property is likewise very hard and is primarily ground through two ball mills followed by grinding in a number of secondary ball mills, one of which has been converted to pebbles using ore. We believe we may be able to save up to 30¢ or 40¢ per ton. The ball consumption two years ago was about 4.8 to 5.1 lb per ton. By studying the ball sizes, ball rationing, and quality of balls, we have reduced that to 2.3. We are using the best ball we can get on account of the freight, but there is still much money to be saved.

I should like to ask Mr. Crocker a question. In the layout at Lake Shore, I believe, the classifiers are situated above the ball mills, so that the discharge from the ball mills is pumped to the classifiers. That simplifies the procedure, but what happens if the classifiers are set alongside the ball mills, preventing the enlargement of the latter, so that for each ball mill now working at 46 pct of its original capacity you have the same classifier that previously handled the whole tonnage?

Harlowe Hardinge *(The Hardinge Co., York, Pa.)*—This paper is very important to the mining industry because it brings out a feature of grinding long neglected. In the early nineteen hundreds nearly all grinding was done with pebble mills. Later, balls were tried. A gain was made in floor space, and a gain in capacity, but very little gain was made in efficiency. However, styles changed, people became interested in doing something different, and the pebble mill was no longer continued in general use in the wet grinding of ores except on the Copper Range in Michigan. We have almost forgotten the advantages of the pebble mill, and I believe Mr. Crocker's paper is valuable if it does no more than bring us back to some of the old fundamentals that were determined many years ago. He has done things somewhat differently from the way they were done then, but his methods are quite similar in some respects to the work in South Africa, where grinding is carried on at low cost. I was there last year and saw operations feeding pieces as large as 10 in. to the mill in the form of grinding media, as well as smaller sizes comparable to those discussed in the paper. This method will create substantial savings in operation costs where it applies, but it does not always apply. It is important in considering what has been done here that we do not assume that pebble mill methods of operation are the same as ball mills to accomplish the same work. By this I have in mind such factors affecting the overall economy of the two methods as mill speeds, pulp densities, liner forms, high or low pulp level discharge arrangements, and mill diameters. We must make calculations by considering these points individually and not try to make too many comparisons between what we would do with a pebble mill to make it operate like a ball mill.

Oscar Johnson *(The Mine and Smelter Supply Co., Denver)*—The Chilean mill in use at the Utah Copper was beginning to give way to pebble and ball mill grinding in 1914. This was about the time that the Marcy Mill was introduced experimentally at the Utah Copper.

Although flotation was known to be successful prior to 1912, the Utah ores were not entirely treated by this process until about 1923. During this period the fine grinding was taken over by the installation of 7x10-ft mills using 2-in. balls as a grinding media. By 1930 all grinding for flotation was done by 7x10-ft primary mills using about 2⅜-in. balls, and secondary mills of same size using 2-in. balls. I well recollect that during this early period the Mine and Smelter Supply Co. handled orders from the Utah Copper Co. for Danish and French pebbles and also secured considerable quantities of cobbed materials from quarries located in Iowa and Minnesota. Using these materials must have been a successful expedient until Utah Copper could get into production making iron balls. Evidently they gave maximum capacity, which was the main objective.

I am very much interested in Mr. Crocker's statement that Lake Shore is using low discharge mills, since in 1937 I interested Mr. A. L. Blomfield in our grate discharge arrangement; he gave us an order for one which I am sure started Lake Shore off on the operation of the low discharge mills. Canadian Institute of Mining and Metallurgy *Bulletin*, Vol. 43, carries an illustration of this low discharge screen with the 8 lifters which we suggested as proper for handling the discharge of the tube mill. I also recollect that when Mr. Francis Bosqui was going to South Africa we gave him data on the grate discharge, thinking he might wish to suggest it to the milling men in Africa. The Utah Copper Co. has applied low discharge grates to their 7x10-ft primary mills.

Bunting S. Crocker (author's reply)—Mr. Myers asked if this process would work on soft ores. Many people have asked that same question. Actually we have not tested soft ores, but we have many reasons for thinking that pebble grinding would work as well or even better with softer ores. The South African ores are much softer than those at Lake Shore, and pebble milling has been successful there for years. Their daily pebble consumption is about ten times ours. The equipment we put in to handle 30 tons of pebbles per day could easily handle 100 to 200 tons per day. Therefore on softer ores the savings on the amount of rock that bypasses the crushers is considerably greater. We save 1½¢ per ton by not having to crush 3 pct of our ore; plants with softer ores might save 4¢ to 5¢ per ton. The pebbles from softer ores also round up more quickly and make more spherical pebbles. There seems no reason why most ores should not be tested by this process. When we started work on this problem there appeared to be thousands of obstacles, but they nearly all resolved themselves; it did not turn out

to be half so complicated as we thought it was going to be.

Mr. Myers next asked if we had further information about correct pebble size. We have done a great deal of work on this subject since the paper was written, and as it is long and rather detailed, I think we will publish it separately in a second paper which will appear in the Canadian Institute of Mining and Metallurgy *Bulletin*. Since it is impractical to screen rock to exactly one size, pebble feeds are always similar to "rationed ball feeds," which is usually an advantage, and it is therefore necessary to make several calculations to arrive at a pebble load in the mill which will have the same weight per average piece as the previously determined correct size of steel ball.

The last question asked by Mr. Myers concerned the upper range of pebble grinding. Mr. Hardinge partly answered this question when he reviewed the South African practice. Over there they use pebbles that are up to 10 in. in size. We have tested pebbles up to 6 in. in size, equivalent to 4-in. steel balls, and are considering replacing our second stage ball mill with a rock mill. We would save an additional 2½¢ on this change. Our present pebble plant is handling 8 and 10 mesh material and I think pebble milling with 2-in. pebbles could be carried to 6 mesh in many plants. I might mention that Wright-Hargreaves has already put this process in and has run it successfully for the past two years. Their saving is 21¢ per ton as against our 28¢ because their ball milling section was stronger than ours and it was not convenient for them to do as much work in the pebble mills. We do 85 pct of our wet grinding in the pebble mills.

Mr. Weiss asked what would happen to auxiliary equipment such as classifiers, motors, etc., if a steel mill were converted directly, thus having only 47½ pct of the original capacity. We solved this point by expanding the diameter of the mill until it drew the same power and had the same capacity as the original mill. Therefore the motors, classifiers, etc., were in balance. The fact that our bowl classifiers were on the upper floor does not really make any difference.

There are two other points that have come up since the paper was first written: 1—Grinding with pebble media eliminates much of the chemical reducing action of iron. With 5x16 steel tube mills grinding in cyanide solution, the oxygen saturation of the solution in the discharge was 9 pct saturated. This is detrimental in cyanide work and we had to use three primary agitators to bring the oxygen back into the solution before thickening. Using pebble mills, the solutions are 67 pct saturated in the pebble mill discharge, and we only use one primary agitator now. Thus the "conditioning", in a chemical sense, that takes place in grinding mills is different with pebble mills from steel mills. Some flotation plants may even have to add fine iron as a reducing agent if that effect is desired. Cyanidation plants of course will benefit. 2—The stage grinding shown in this report gives a better sulphide distribution for flotation than the previous two-stage circuit. We now get less fine pyrite and less very coarse pyrite in the flotation feed. Most of the pyrite is in the 20 to 40 micron sizes, which is ideal for flotation recovery.

DISCUSSION

L. E. Djingheuzian *(Canadian Department of Mines and Technical Surveys, Ottawa)*—This paper summarizes the results of exceedingly useful work done at Lake Shore and presents some unusual data. By analyzing these data, I have obtained a few tentative figures which I believe might be of interest.

Before Lake Shore changed their fine grinding with steel to fine grinding with screened ore as pebbles, their steel consumption was 4.3 lb per ton of ore. After the changeover to grinding with pieces of ore as pebbles, the pebble consumption was determined to be 28 tons per 1000 tons of ore, or 56 lb of pebbles per ton. From this, the ratio of pebble consumption to steel consumption by weight works out at 13 to 1.

Total gross power input in fine grinding of 1000 tons at Lake Shore is (182x8) = 1456 hp, or 26.1 kw-hr per ton. Assuming 20 pct frictional losses in the mills and the useful energy input from tumbling media in grinding hard siliceous ore at 15 pct, according to Schellinger's experiments,[10] the useful energy input per ton of ore at Lake Shore will be 12 pct of the energy input to the motor, or 3.13 kw-hr.

Since Lake Shore screened ore wears 13 times as fast as steel, the net energy required to wear a ton of steel, formerly used at Lake Shore, will be:

$$3.13 \times 13 = 40.69 \text{ kw-hr,}$$
$$\text{or } 161.7 \text{ joules per g steel.}$$

Since for unit weights the ratio of the surface of Lake Shore ore, sp g 2.70, to the surface of steel, sp g 7.80, is 3.094 to 1.525, the same volume of steel absorbs in grinding to the same mesh 13x3.094 to 1.525 as much energy as the same volume of Lake Shore ore.

The amount of surface energy stored in grinding quartz was determined by Schellinger[10] as 107,000 ergs per cm^2. Assuming that Lake Shore ore stores the same energy in grinding as quartz, the energy stored in steel in grinding will be

$(107{,}000 \times 13)\ 3.094 \text{ per } 1.525 = 2{,}821{,}900 \text{ ergs per } cm^2$

I would like to know how this figure checks with any determinations, if ever made, by physical metallurgists on surface energy stored in steel during work hardening before rupture.

Using this figure, the amount of new surface formed in grinding of 1 g of steel to 90 pct —325 mesh, Lake Shore grind, will be

$$(161.7 \times 10^7) \text{ per } 2{,}821{,}900 = 573 \text{ cm}^2$$

[10] A. K. Scheilinger: Solid Surface Energy and Calorimetric Determinations of Surface Energy Relationships for Some Common Minerals. *Trans.* AIME (April 1953), pp. 368-374.

The Third Theory of Comminution

by Fred C. Bond

DISCUSSION

Milton C. Shaw and **Donald R. Walker** *(Massachusetts Institute of Technology)*—For many years the two empirical rules of comminution bearing the names of Rittinger and Kick have been applied to grinding processes to account for the energy required to crush brittle materials. The basis for Rittinger's rule was the thought that the energy to grind a substance went largely into supplying the additional energy associated with the new free surface, while Kick's law was prompted by the concept of dimensional similarity and is a special statement of the concept of constancy of strain energy to cause rupture. According to Rittinger the energy to grind a given volume varies as the change in the square of the mean particle size, $v \sim \Delta(D^2)$, while according to Kick this energy varies

as the change in the cube of the mean particle size, $v \sim \Delta(D^3)$. These two rules at first appear to be incompatible, and considerable controversy is to be found in the literature concerning which of the rules is the true one.

The present author has contributed to this controversy by proposing that a rule intermediate between that of Kick and Rittinger be adopted. The energy required to grind a given volume of material is assumed to vary as the change in the 5/2 power of the mean particle size, $v \sim \Delta(D^{5/2})$. While it is inferred that a derivation for this intermediate relationship is presented, it would appear to be more accurate to refer to it as an assumed relationship, since it is not based on a clear-cut model or physical picture.

In more recent times it has become clear that a negligible part of the energy in comminution is associated with the production of new surface. Grinding efficiencies, based on surface energy, of the order of but 0.1 to 1 pct are normally reported. It is thus evident that the physical basis for Rittinger's rule is grossly in error. However, we are left with the fact that there is considerable experimental data in essential agreement with the Rittinger relationship. It would then appear that what is needed is a new physical basis for the old rule.

In recent metal cutting studies[10] the energy required to remove a given volume of metal was found to increase as the particle size was decreased. This trend was found to continue to a certain critical size, corresponding to a layer about 1 micron in thickness, beyond which the energy per unit volume remained constant.

All materials contain numbers of imperfections and inhomogeneities which affect the amount of shear stress necessary to yield or rupture the material. As the volume of the material subjected to stress is reduced, the probability of finding an inhomogeneity to act as a strength-reducing agent is decreased. The material appears to have higher strength than it does in bulk and the energy required to remove a given volume thus increases. When the energy ceases to rise with decreasing particle size, the material is exhibiting the theoretical strength of a perfect lattice structure, since it is being forced to shear in regions between inhomogeneities. The region in which the energy per unit volume is inversely proportional to the particle size is in full agreement with Rittinger's rule, while Kick's law precisely expresses the observed energy behavior in the region where full theoretical strength obtains.

It would thus appear that Rittinger's rule should hold for relatively large particles, larger than about 1 micron, while Kick's law should hold for smaller particles. The literature reveals that in the main the proponents of Kick's rule were dealing with fine grinding while the champions of Rittinger were engaged in relatively coarse grinding. It is to be noted that while each of the two rules appears to satisfy experimental data reasonably well, the reason they do is entirely different from the reasons originally advanced by Rittinger and Kick. The basis for the existence of two empirical rules is the size effect in materials associated with flaws and inhomogeneities. If materials had a single valued strength regardless of size, Kick's law would always hold. In practice Kick's law ceases to hold when the particle size becomes great enough to include inhomogeneities. It is a pure coincidence that in this latter region Rittinger's rule, which was originally derived from a false physical picture, is in general agreement with experimental results and the concept of a variation of particle strength with size.

While the aforementioned reasoning is based on results for ductile metals, it must be realized that brittleness is also a function of particle size. Materials are brittle only because they contain inhomogeneities sufficiently large to prevent them from being ductile. In crushing materials normally considered brittle in bulk, the major portion of the energy is associated with plastic flow. The observed negligible difference between the energy required to crush materials normally considered to be ductile and brittle is a further consequence of the size effect in materials.

Supporting data and a fuller treatment of the ideas set forth in the above discussion will soon be submitted in a paper on the theory of comminution.

[10] W. R. Backer, E. R. Marshall, and M. C. Shaw: The Size Effect in Metal Cutting. *Trans.* A.S.M.E. (1952) 74, p. 68.

R. J. Russell *(Hardinge Co., York, Pa.)*—The paper as originally written for presentation in Mexico City and as revised states that the Rittinger theory is not tenable mathematically because of the presumed lack of consideration of the distance factor. It also lists three assumptions presumed to be essential to the development of the Rittinger theory.

The Rittinger theory states simply, "The energy necessary for reduction of particle size is directly proportional to the increase of surface." A practical consideration does not require and cannot have a datum point of theoretical zero surface area.

A simple means to illustrate the Rittinger concept has been the breaking of a cube into smaller cubes. It is not required that this be done by compression. It is unimportant in the Rittinger concept how it is done. It may be by shear or tension. Regardless of the manner in which fracture is obtained, the deformation before fracture provides a distance factor and the Rittinger theory is not indefensible mathematically.

Attempts have been made to correlate work done in grinding to surface energy, but this is no part of the original Rittinger concept, even though it may be a valid concept.

It has not been proved in the paper that the Rittinger theory is not correct.

Recently there is a tendency to consider the Rittinger theory a too simple concept of a complicated phenomenon. Any lack of consideration on grounds of simplicity rules out logic. When any material is subdivided, surface is increased, but neither mass nor volume is changed. It is logical, then, that a relationship to surface exists.

During the past 20 years several competent investigators of crushing and grinding processes have provided confirmation that the Rittinger theory is substantially correct. The work of Gross and Zimmerley demonstrated the validity of the Rittinger theory, and despite the problems encountered in measuring surface areas other observers produced further substantiation.

In 1938[11] Bond and Maxson stated that their work on the grinding characteristics of ores conclusively establishes that "the useful work done in grinding is proportional to the new surface area produced; in other words, Rittinger's law is the correct expression of work done."

Difficulties encountered in obtaining absolute measurements of surface area do not in any way invalidate the Rittinger theory. The practical application of Rittinger is limited only if relative surface area data cannot be established, and these are as accurately and readily capable of establishment as diameters which must be approximated before application of formulae developed in the paper.

Assumptions no less questionable than those cited against Rittinger are made by the author in the derivations offered as well as the confirmations. Substantiation of Rittinger by tests on quartz ore are discounted, yet the author offers a confirmation based on crushing quartz crystals and glass spheres.

The Rittinger theory implies that size distribution of any given feed is unimportant so long as surface is known, and the same is true for the product. Therefore, tests made which confirm Rittinger and use a screened or sized feed are valid so long as areas of feed and product are known or related.

The Third Theory states that work useful in breakage is inversely proportional to the square root of the diameter of the product particles. If this is true, a scalped or sized feed should require no special treatment so long as particle diameters are determined.

Presumably the equivalence of a normal feed is determined in the section on Effect of Scalping Feed, and if so the work index value should not change. In fact, Mr. Bond states: "The work index is constant over all size ranges for a homogeneous material reduced at the same relative mechanical efficiency."

It is my opinion that the tabulated data from the laboratory tests and plants do not provide any proof of the stated Third Theory. These data are based upon empirical relationships established by Mr. Bond and as such may have practical application in approximating equipment size and power requirements for certain crushing and grinding requirements. Mr. Bond calls attention to the necessity for applying properly a judgment factor. This limits practical use of the procedure to benefit only those having experience in crushing and grinding.

Since according to the Third Theory the useful work is directly proportional to the square root of the surface area formed, an experimental proof should be established from surface area determinations. If this were to be done it would necessarily expose errors in work which has been done and which appears to confirm the Rittinger theory.

[10] Fred C. Bond and Walter L. Maxson: Grindability and Grinding Characteristics of Ores. *Trans.* AIME (1938) 134, pp. 296-317.

Fred C. Bond (author's reply)—The statement by Professor Shaw and Mr. Walker regarding Rittinger and Kick can be clarified to read: "According to Rittinger the energy to reduce a given particle to product particles of constant size and similar shape varies as the square of the given particle diameter, while according to Kick it varies as the cube of the given particle diameter." According to the present author the exponent should be 5/2. Under the Rittinger concept the useful energy input goes into the formation of new surface area, under the Kick concept it goes into the reduction of particle volume, and under that of the present author it goes into the formation of crack tips, which extend through the particle, resulting in breaks. Each of these three concepts is fundamentally an assumed relationship, and the correct one should be selected by its conformity with laboratory tests and commercial crushing and grinding installations. Other things being equal, the most accurate comparisons are those which involve the largest range of particle sizes.

The second form of the Rittinger theory is a logical development of the first form, and the commentator's statement: "It is thus evident that the physical basis for Rittinger's rule is grossly in error," which refers to the second form, is warmly applauded.

Inhomogeneities probably exist in practically all the materials subject to crushing and grinding. However, these inhomogeneities extend down to the supposed micelle structure at about one micron, far finer than the ordinary range of fine grinding, and the theoretical breakage of a perfect lattice structure has little bearing on ordinary crushing and grinding problems. Breakage cracks probably follow the micelle boundaries, and the effects of coarser inhomogeneities vary greatly with different materials, depending upon their relative strength, arrangement, and spacing. The coarse inhomogeneities are far too irregular and variable to permit the formulation of a rule covering their effect on the crushing and grinding of all materials. In fact their effect appears to be of little importance in many instances, especially in grinding. So-called natural grain sizes show up in the size distribution curves of some materials but are apparently completely absent in many cases.

In the literature of comminution it has generally been reported that the Kick theory agrees more nearly with the actual energy required in coarse crushing where the coarse inhomogeneities are presumably more effective, and the Rittinger theory with grinding. But neither theory agrees well with the facts when reduction over a large size range is considered.

The behavior of brittle materials under stress is very different from that of ductile metals because of the difference in strain energy flow after initial failure, and analogies between the manner of breakage of brittle rock and that of ductile metal are apt to be misleading.

Mr. Russell defends the Rittinger theory in its original form, but offers no defense of the application of the surface energy concept to it. The original Rittinger theory states that the energy necessary for reduction of particle size is directly proportional to the increase of surface, while according to the Third Theory it is directly proportional to the square root of the increase of surface. The numerical conflict between the two theories is whether the exponent of the new surface area should be 1 or ½.

However, there is a wide difference in the mental processes by which the theories were derived. The Rittinger advocates consider that the production of new surface is the direct object and the criterion of the work done in comminution, while the Third Theory implies that the production of active crack tips, which are extended and split the rock by the resident strain energy of the deformed body, is the immediate object and proper criterion of the work done. The Rittinger theory requires the measurement of surface areas alone, and entirely neglects the deformation or strain distance necessary to produce the new surface, thereby denying the definition of work as the product of force and distance. The work input can be calculated directly from easily obtained size distribution analyses by the Third Theory, while the Rittinger theory requires difficult, inaccurate, and often impossible surface area measurements.

John Gross states:[11] "Several investigators have determined the absolute efficiency of crushing on the assumption that the energy required in crushing is only that necessary to create new surface energy. Such calculations are of academic interest only and have no practicable significance, as they ignore the distance through which the crushing force must act before new surface is produced."

The use of any comminution theory without the concept of theoretically infinite particle size, or zero surface area, is analogous to the use of a portion of a thermometer scale on which the zero reading is unknown. The total energy input can be compared to the absolute temperature, which made the entropy concept possible.

The primary objection to the illustration of the theoretical breakage of cubes to demonstrate the Rittinger or Kick theory is that the shape is particularized; no matter whether compressive, tensile, or shearing forces are used. The cubical shape and the even distribution of stresses do not correspond to actual crushing and grinding conditions.

The experimental confirmations of the Rittinger theory by Gross and Zimmerley and others suffer from at least two defects which did not appear important when the theory was tacitly accepted before the tests were made, but which appear larger when it is challenged. The first of these is that the range of size reductions compared was so limited that the difference caused by the surface area exponents of 1 and ½ would be small and might be disregarded. It is only too natural to accept results which confirm what is expected and believed, and to reject readings which do not exactly conform as erroneous, especially when those readings cannot be explained by any known hypothesis. At present, comparison of the results of crushing to 1 in. with grinding to 200 mesh should confirm the Third Theory rather than that of Rittinger.

The second defect is that most of these tests were made with an artificial feed composed of particles of only one screen size, in order to simplify the surface area determinations. It is true that this should make no difference if the Rittinger theory is previously accepted as being correct, but it can make a large differ-

ence if the Third Theory is correct. The present author made this error in the 1938 paper,[13] in which −14 +20 mesh screened particles were reduced in the pendulum impact device to measure the surface area produced by impact, and results were obtained which appeared to substantiate the Rittinger theory.

The work index is valid only for a natural feed containing the normal fines resulting from crushing or grinding; the section headed **Effect of Scalping Feed** describes a method for obtaining the approximate natural feed size when a scalped feed of known size distribution and known hourly tonnage is used. In most cases where a crusher feed is scalped by passing through a grizzly ahead of the crusher it is more satisfactory to disregard the action of the grizzly entirely, since the hourly tonnage and size distribution of the material removed by the grizzly are usually not known.

When the plotted percent passing size distribution lines of the feed and product are parallel, the reduction ratio remains constant for all sizes, and the work index calculated from the convenient 80 pct passing size is equivalent to that calculated on the basis of any other selected passing size. If the lines are not parallel the work index will be affected, but the effect will be relatively small for small differences in slope of the plotted lines, and will be only the square root of the effect on the new surface area produced. However, scalping the feed produces a large effect on the work index which requires correction, and using feed particles of a single screen size requires a still larger correction.

Variations in the ratio of interior to outer surface in crushed quartz particles of different screen sizes may account for differences in the work indexes at these sizes. John Gross in Crushing and Grinding, Table 14,[14] shows that this ratio varies from 2.64 for 3x4 mesh particles to 0.11 for 150x200 mesh. Glass marbles have no interior surface.

Laboratory grindability tests yield work indexes which are valid only over the size range where the machine operates efficiently. When a hard ore is reduced in a fine grinding ball mill the efficiency begins to drop off with feed sizes larger than about 6 mesh, and such a mill fed with ½-in. ore will require more work input per ton than that calculated from the grindability test. Correction factors for oversize feed and other operating variable must be calculated from experience.

Many additional confirmations of the Third Theory which are not described in the paper have been obtained from commercial crushing and grinding installations, the most striking feature of which is the general agreement of the work index calculated from crushing plant results with that calculated from fine grinding and intermediate reduction stages in the same plant. Since the Rittinger theory is notoriously inconsistent in comparisons over a large size range the author is convinced that the Third Theory agrees better with the facts of comminution.

[14] John Gross: Crushing and Grinding. U.S. Bur. Mines *Bull.* 402, p. 7.

Flotation Rates and Flotation Efficiency

by Nathaniel Arbiter

DISCUSSION

Gregoire Gutzeit and **Porter Clements** *(General American Transportation Corp., Chicago)*—We find Professor Arbiter's paper especially interesting in view of rate studies we have conducted in our laboratories in connection with the recovery by flotation of fine coal from mine washery waste waters and river deposits. In an attempt to evaluate the effects of design and operating variables of a flotation machine developed by our company specifically for coal and for the purpose of scaling-up laboratory batch flotation results for plant design, an approach was made through a study of flotation kinetics.

Investigation of the (combustible) recovery-time variables showed that a plot of time/recovery vs time gave the simplest linear representation of the data, tests being conducted by preliminary conditioning with collector, followed by flotation with stagewise reagent addition. It is interesting to note that in this case, where the bulk of the feed is floated, the data may be graphically represented in the same manner as the data shown by Professor Arbiter for ores containing a relatively low proportion of floatable minerals.

In comparing data from two batch coal flotations run in the same laboratory machine by preparing t/R vs t plots and determining rate constants by extrapolation, Professor Arbiter's method is found to be extremely sensitive, since the t/R intercept is usually less than 1. A small difference between intercept values in this range becomes magnified when the reciprocals or A^2K values are calculated. For instance, two different anthracites, 28x200 mesh, of about the same ash content were batch-floated in the same laboratory machine under approximately the same conditions for 2.5 min, giving combustible recoveries of 0.936 and 0.976. The t/R vs t plots were reasonably linear. A^2K values were 1.5 and 4.1, respectively. It should be mentioned, however, that the first two points for each set of data, 0.25 and 0.50 min, fall above the line, probably because of incomplete conditioning, hence, lower initial recovery and higher t/R values. Unless all of the plotted points fall exactly on a straight line, the choice of the most representative line is at the discretion of the experimenter. As shown above, for almost the same average or overall rates, A^2K values differ considerably, the sensitivity being inversely proportional to the value of the intercept. Unfortunately, in the case of coal flotation, t/R vs t plot, the area of greatest importance to the extrapolation is characterized by somewhat scattered points. It would appear, then, that the use of this rate constant to evaluate floatability or flotation machine performance would have greater significance for the slower-floating minerals. However, the positions of the first few points seem to give a good indication of the effectiveness of the conditioning step.

In deriving a theoretical flotation rate equation, Volkova[15] attempted to show the influence of relatively nonfloatable material on the mineral being separated. His integrated rate equation contained both first and second order terms, having the form

$$Kt = (1\text{-}Z)\left(\frac{R}{1\text{-}R}\right) + Z \ln\left(\frac{R}{1\text{-}R}\right)$$

where K is the rate constant, t the time, Z the fraction of more floatable mineral initially present in the feed

solids, and R the recovery of the mineral based on a maximum possible recovery of unity. It is indicated that for small values of Z the second order term would be dominant, while for large values or for the flotation of a single mineral ($Z = 1$), the first order term would be significant. Recovery-time data presented by Arbiter for ores where Z was small appear to give the second order hypothesis some foundation from the resulting linearity of the t/R vs t plots and from his Table II showing the dependence of recovery upon initial mineral concentration. It should be pointed out, however, that the presence of the same variable, time, in both terms tends to mask the true nature of the R-t function; since the second order expression is rearranged and plotted $\frac{R}{1-R}$ vs t, the points do not fall as close to a straight line. If Volkova's R-t data, calculated from the above equation for three barite-quartz mixtures, where Z was 0.2, 0.3, and 0.5 for barite, are plotted both as t/R vs t and $\ln \frac{1}{1-R}$ vs t, the latter graphs are found to be quite sensitive to changes in Z, giving rather widely separated nonlinear curves. The t/R vs t plots result in closely-spaced curves with linearity increasing with the increase in Z, a trend in opposition to what should be expected. The plot for the 0.5 barite mixture is perfectly linear, again showing the ability of the $\frac{t}{R}$ vs t plot to represent rather than characterize the data.

Sutherland's R-t data for glass spheres[3] and our own coal flotation results do not show linearity in directly plotting $\ln \frac{1}{1-R}$ vs t, but equations containing these terms may be made to fit the data by introducing constants to achieve linearity. Sutherland developed an equation fitting his glass sphere R-t data with an accuracy of ± 1 pct, but his method does not constitute a practical approach to the problems of evaluating floatability and flotation machine performance.

To summarize, we believe that since any one of several types of equations can be made to conform to a set of R-t data by the incorporation of suitable constants in the chosen form, the multitude of variables influencing flotation rates will continue for the time being to resist efforts to describe the mechanism by a single generalized equation.

For the present, we are forced to consider flotation as a rate process of unknown order, basing our calculations and conclusions on graphical analyses of the R-t data.

[2] Z. Volkova: *Acta. Phys. Chim. U.R.S.S.* (1947) **22**, p. 331.
[3] K. L. Sutherland: *Journal of Physical and Colloidal Chemistry* (1948) **52**, p. 394.

F — Coal

The Formation of Acid Mine Drainage

by Kenneth L. Temple and Arthur R. Colmer

DISCUSSION

E. W. Lyon *(Bureau of Mines, Pittsburgh)*—Previous to the first World War James Campbell and E. C. Auld of the H. C. Frick Coke Co. carried out a successful large-scale pilot plant experiment on the treatment of mine drainage and patented certain features of the process. The writer's first serious contact with the acid mine drainage problem began when he was Managing Engineer for this plant and has been more or less continuous until the present time. As early as 1919, Powell and Parr in Bulletin No. 111, University of Illinois Experiment Station, indicated that biochemical action was probable in the oxidation of coal seam sulphides to sulphates in the absence of air. During the twenties and early thirties, Sayers and Leitch of the Bureau of Mines, with Van Zant and others of the Sanitary Engineering Bureau, Pennsylvania Department of Health, developed the principles of the method of abatement later known as mine sealing. Between 1935 and 1942, engineers of the U. S. Public Health Service and eight cooperating State Health Departments conducted a vast full-scale experimental application of the mine sealing process at 4000 mines, from Indiana to Maryland and from Pennsylvania to Alabama. Nearly all these workers agreed, in private conversations, that the reactions taking place in the mine were more variable and more complex than had been anticipated. Their repeated recommendations for specialized studies have motivated the current investigations which, while exploring new channels, confirm many earlier concepts.

The authors of this paper, Dr. Temple and Dr. Colmer, have contributed something new and vital to mine drainage technology. For this they are to be highly complimented. They have also made some statements, it is believed inadvertently, which are sure to arouse disagreement.

The introductory paragraph contends that acid mine drainage presents a "peculiarly difficult problem for two principal reasons."

1—The first is that the acid discharge increases in proportion to the expansion of the industry. In this respect the mining industry is not unique, since expansion is common to most pollution sources, both domestic and industrial. It is unique in that the pollution persists, or even increases, after mining operations have ceased and responsible ownership has abdicated. In justice the individual operator cannot be expected to underwrite a perpetual control, or do more for his drainage than is being done for the abandoned mine drainage. The difficulty here is political, and lies in establishing the equity between private and public responsibility.

2—It is stated secondly that "there is no method yet known that will appreciably alleviate the acid water problem except at totally prohibitive cost to the coal operator." On the contrary, it may be shown that there are two methods which will provide substantial improvement. These are chemical treatment and mine sealing.

Chemical treatment is admittedly expensive, but if full advantage is taken of present knowledge it is not necessarily as unthinkably expensive as many have claimed. In the upper Ohio River Valley the mines discharge approximately 3 million tons of acid per year—equivalent ton for ton to calcium carbonate or limestone. Over the same area the damages due to acid drainage have been variously estimated at $5 million to $30 million annually. At the present value of the dollar it is probably nearer the latter figure. It would be reasonable, then, to assess at least $5, and perhaps as much as $10 million dollars, against each ton of acid. In computing the total cost to the industry, it has usually been assumed that it would be necessary to treat all the acid. However, a large portion of the total imposed load on the streams is neutralized by the natural alkalinity of the surface runoff. Surveys also show that nine tenths of the acid load is discharged from about one third of the mines. The average production from this group of mines may be 700 to 1000 tons of acid per mine per year. If properly handled, about $3.00 worth of pulverized limestone and $3.00 worth of hydrated lime will neutralize each ton of such acid. Under certain favorable conditions, it is possible to recover both iron oxide and aluminum oxide in commercial quantities and possibly other byproducts. It may be deduced, therefore, that giving nature assistance at particular trouble spots may well be practical. Chemical neutralization of mine drainage is not a cureall, and is not universally applicable, but it has a place when used with discretion. Its main disadvantage, other than cost, is that most of the sulphate remains in solution as permanent hardness, forming compounds objectionable to water users.

The combination process of air and drainage control known as "mine sealing" has been demonstrated throughout eight states by a $10-million experiment carried out under the sponsorship of the U. S. Public Health Service and the several State Health Departments. Notwithstanding that the actual work was done under the limitations imposed by the use of relief labor, the favorable results were too widespread and too well authenticated to be disregarded.[4] As was to be expected from the state of the art, efficiency varied greatly from location to location. On the average, the overall discharge of acid sulphate salts from sealed mines was reduced 70 pct, usually accompanied by a simultaneous reduction in volume of drainage. Unfortunately, most of the operating data is still buried in the archives. Because of the restrictions imposed on the expenditure of public money, the program was limited to abandoned mine workings where it was considered that there was no responsible owner. The system is nevertheless readily applicable to worked-out sections of operating mines. If the mine projection has provided for sealing successive panels, the cost is very moderate and may frequently result in an overall saving due to reduced ventilation and drainage costs. Since over 80 pct of the total acid discharge comes either from abandoned mines or worked-out portions of operating mines, the remainder from truly active areas need give little concern. It can scarcely be said, therefore, that "there are no methods yet known that will alleviate the acid water problem except at exhorbitant cost. . . ." The fact that a process is not perfect does not preclude its being used until it can be improved or a better method found.

The contention that the roof shales and rider coals constitute the prime source of acid-forming minerals is most pertinent, and the observation that there is no substantial acid production until the strata is fractured and exposed to the mine atmosphere is equally valuable. The scientific determination and documentation of these facts is probably an even more useful contribution than the discovery that a specific form of bacteria is actively associated in the biochemical chain of reactions. Engineers engaged in the underground phases of mine drainage have long noted and verbally discussed the fact that acid conditions are not bad during first mining. Pipe lines from gathering pumps at the room faces are little affected. After pillar robbing has fractured the overburden the corrosivity and volume of the drainage both increase greatly. Thus these unrecorded observations support and extend the findings of the authors.

With regard to the sulphur oxidizing bacillus, the authors do not state whether an acid condition is first necessary for the growth of the bacteria or whether the bacteria can be found on neutral or alkaline surfaces containing sulphur compounds which they thereafter decompose with the production of acid. It is implied that moisture is necessary, but it would be interesting to know if the bacteria is aerobic or anaerobic, that is, whether or not the bacteria can exist and multiply if the sulphur-bearing slate is completely and continuously submerged in stagnant water. Other obvious questions might be asked, but it is presumed that the authors are already working on them.

In view of the statement that the source material producing mine acid has not been exactly identified, it is believed that more proof should be awaited before discounting the presence of free acid. It is true that there are many analyses in which the metallic ions are equivalent to the sulphate ions, but there are also many analyses that can only be balanced by counting the free hydrogen ions as evidenced by the pH determination. Theoretically the pyrite, iron sulphide FeS_2, should decompose to ferrous sulphate, $FeSO_4$, and sulphuric acid, H_2SO_4. As the iron changes valence by oxidation from ferrous to ferric ions, a portion of the free acid is recombined, $Fe_2(SO_4)_3$, to satisfy the added sulphur demand of the iron. Other portions of the free acid may dissolve aluminum compounds in transit and appear in the analysis as aluminum sulphate. If sufficient calcium carbonate is encountered in passage, all of the actively ionized acid is combined and will be found as calcium sulphate. From a variety of analyses it appears that any one of these conditions may prevail. There may be much free acid or no free acid, depending upon time and place. Incidentally, there is a surprising uniformity in the sulphate ratio in the mineral analysis of mine drainage which bears out the above presumption. Whether the particular discharge is acid, neutral, or alkaline, the sulphate radical, $SO_4^{=}$, remains at a remarkably constant percentage of the total mineral content.

It is submitted that the load of polluting salts conveyed by mine drainage may eventually be traced to a leaching action upon the freshly broken rock, a rapid decomposition of iron sulphides by bacterial action as indicated in this paper, and a slower but possibly more sustained decomposition of the more stable pyrites and marcasites through chemical oxidation. The new evidence currently presented may be interpreted as confirming the main tenets of mine sealing, as developed and amplified through the demonstration program of the U. S. Public Health Service, and it is believed that mine sealing still offers the most promising, reasonably effective means of abating the nuisance at low cost.

We do need to know much more about the detailed behavior of acid mine drainage under varying conditions from earliest formation to final disposition before maximum efficiency and economy can be attained. It is to be presumed that an amount equal to the damages caused by acid mine drainage could be profitably expended to eliminate the damage. Such figures, the result of a thoroughgoing impartial appraisal by experienced valuation engineers, are not available. It would seem that any serious attempt to solve the

problem should begin with a determination of this factor. Secondly, it is fairly certain that any method, mine sealing for instance, if applied to a mine would modify its layout and operating schedule, maybe for better and maybe for worse. Here, not research in its proper sense, but just plain engineering-management studies and experimentation on full-scale operating mines is needed. Just what is the effect, at representative mines, upon costs, drainage, ventilation safety, panel sequence, corrosion of rails and other equipment, and similar factors? The answers might be startling, either for or against the proposal. Prior to the determination of such facts it is quite probable that whenever the industry, either directly or indirectly through public agencies, expends upon mine-drainage-stream-pollution as much as 5¢ per ton of coal mined, the problem will no longer exist.

[4] Ohio River Pollution Control, House Document No. 266, 78th Congress, 1st Session.

Kenneth L. Temple (author's reply)—Mr. Lyon is quite correct in pointing out the contributions of earlier workers in the field of acid mine drainage. A complete list would contain many more names. Frank Haas, a practical mining man who was prominent in the development of the northern West Virginia coal field, had a deep interest in acid formation and a remarkably clear insight into the problem. The first accurate biological studies of acid water were carried out by J. B. Lackey. Many workers at West Virginia University and elsewhere have devoted much time to a study of acid mine drainage, its formation or treatment. In addition to those mentioned by Mr. Lyon, the names of L. V. Carpenter, L. K. Herndon, W. W. Hodge, D. B. Reger, E. T. Reetman, W. R. Downs, A. H. Davidson, and G. L. Hall are associated with earlier studies of acid drainage. In spite of the importance of the problem as indicated by these investigators, no full time studies were made until the inception of the BCR project at West Virginia University and the Mellon Institute project. For a short period there was also a full time project at the U. S. Bureau of Mines in Pittsburgh.

Mr. Lyon in his criticism of the opening paragraph stressing the importance of mine water pollution apparently missed the point, which is that since worked-out mines continue to make acid, the expansion of mining steadily decreases the amount of surface water that is free from acid mine drainage.

Chemical treatment is feasible only in the milder examples of acid mine water. It is not practical by any known method for the highly acid waters so widespread in the great bituminous coal area of northern West Virginia, southwestern Pennsylvania and eastern Ohio and of occasional occurrence elsewhere. The best study of chemical treatment applicable to the American soft coal fields is that by S. A. Braley of Mellon Institute in a report entitled *A Pilot Plant Study of the Neutralization of Acid Drainage from Bituminous Coal Mines*, published by the Pennsylvania Department of Health.

The mine-sealing program still has its adherents. Perhaps a good study of the results of the mine-sealing program should be made, if only to provide the data for settling arguments about its merits. Few of the early follow-up reports on mine sealing are worthwhile, inasmuch as the mines were still filling at the time of the reports and the low acidity measurements made at that time cannot be duplicated today. It is the opinion of the authors that mine sealing, so-called, as applied to the slope or drift mine, is an expensive and ineffectual procedure.

The question whether an acid condition is first necessary for the growth of the sulphur-oxidizing bacteria is practically meaningless. The reason is the small scale of the bacterial world. An acid spot accidentally produced, a few square microns in area, would be sufficient to allow one bacterium to start growing. The acid formed by this bacterium would spread out and make the surrounding area acid enough to allow more cells to grow. In order to inhibit bacterial growth through control of acidity it is necessary to have a buffered solution bathing all sulphuritic particles. This, of course, is not the case in the mine. Since the present paper deals primarily with the source of sulphuritic material from which acid is formed, the many aspects of bacterial action are not dwelt upon.

The statement that there is no free acid is based entirely upon analyses by M. E. Hinkle, formerly of this laboratory. His data seem to be conclusive for the type of waters studied, which include the most highly acid mine waters discharged from coal mines. The result is a balance between cations and anions, not including hydrogen ions, indicating that there is no net free sulphuric acid. The high titratable acidity is due to the hydrolysis of various salts, the principal one being ferrous sulphate. It is not inconceivable that samples of acid mine drainage containing some free sulphuric acid might be found; they have not been found in the local coal fields, and would not be expected under the local geological conditions. Perhaps pyrite in sandstone would give such a water sample containing ferrous sulphate and sulphuric acid, but in the authors' experience the water from a mine that has a roof of sandstone containing pyrite is only mildly acid and does not constitute the main problem.

A. A. Berk *(Bureau of Mines, College Park, Md.)*—The Bureau of Mines has been privileged to follow the West Virginia University work on coal-mine drainage through frequent progress reports. This summary paper by Temple and Colmer is based on careful observation in the field and painstaking research in the laboratory. Comments on this work by others in the field emphasizes the essential complexity of the problem, since the interpretation of the data is questioned and acceptance of the observations as fundamental data is disputed.

There is no doubt that Thiobacillus Thiooxidans may increase the rate at which sulphuric acid is formed from sulphides in roof strata, debris, and residual coal in the mines. Such chemical reactions generally occur in several steps and the overall rate is determined by the rate of the slowest step in the chain. The bacteria appear to accelerate the formation of acid by speeding up the rate-controlling reactions. A laboratory demonstration of this effect, including rate determinations, would make an excellent problem for a doctor's thesis at the University.

On the other hand, there is ample evidence that the formation of sulphuric acid does not require bacterial assistance. Elaborate tests at West Virginia University confirmed this basic fact. Thus, although the work of Temple and Colmer is a long stride from the observation of Carpenter and Herndon[4] 20 years ago that Thiobacillus Thiooxidans thrived in sewage made acidic by acid-mine water, it does not appear to present a practical method of coping with the pollution problem.

The one practical suggestion made by the authors is that coating an acid-producing surface to seal it from contact with air would stop acid formation. Such local sealing is impractical both because of the nature of the surfaces and because interference with the movement of seepage water would break surface seals quickly. In contrast, sealing off the air of abandoned mines, or abandoned areas of mines, with water-filled concrete traps should have the same desirable effect. Sulphuric acid formation is known to have been greatly decreased and even stopped in instances in which complete sealing was effected. The apparent success of the energetically conducted mine-sealing program in Pennsylvania is, however, no confirmation that bacteria are needed for the formation of sulphuric acid, since oxygen is a primary factor, without which the huge tonnages of sulphuric acid cannot be formed by any process.

Mine sealing is still the only practical method by which the output of naturally formed sulphuric acid can be reduced. Tonnage reductions achieved in Pennsylvania's program are most impressive. In many localities the output already can be tolerated because of the natural alkalinity of the watershed. Conditions should improve, as sources of incoming air, many located in adjacent active workings, are sealed. However, some form of neutralization or acid removal may be required before tolerably clean streams are restored in the coal country. A solution acceptable to both operators and State Authority is being sought by the Stream Pollution Research Laboratory of the Bureau of Mines.

[9] West Virginia University Research Bulletin No. 10, 1933.

Filter Cake Size-Consist and Moisture Relationships

by O. R. Lyons

DISCUSSION

J. D. Clendenin *(Bureau of Mines, Anthracite Research Laboratories, Schuylkill Haven, Pa.)*—In this paper and earlier ones, Mr. Lyons is to be commended for endeavoring to place coal dewatering on a more rational basis, thereby systematizing available information and providing better tools for estimating design or evaluating performance. The task requires much more effort than is generally appreciated. His contribution would be much more useful, however, if the spread in the relationship shown between average particle size and moisture content of the cake could be reduced, i.e., if he would present available data on other factors influencing the filtering operation and determine their effect on the relationship illustrated in his Fig. 1. In the present paper, the author appears to have abandoned consideration of some of the factors* he endeavored to appraise previously[3, 4] yet there is mounting evidence, as he has suggested, that satisfactory results in dewatering require attention to principles and details.

* For example, the presence of clays and slimes with their great power of water retention in coal filter cakes was assayed by determining the ash content of the —200 mesh material; surface moisture was plotted against the product of percent —200 mesh and the percent ash content of this material. Can details like the presence of clays and other subversive material be neglected? The presence of these subversive materials in fine-coal circuits is a common source of dewatering and cleaning problems. Their removal at the best spot is difficult. If feasible, however, suitable attention to pH may greatly assist[15, 16] in controlling and minimizing water retention by the filter cake, either by removal of these fines and slimes early enough or by converting them into a less water-retentive form. One objective many operators strive for is the removal of these materials before they enter the fine-coal circuit.

On the basis of the evidence presented here, I question, at least in part, the author's conclusions. For example, at average particle size 0.0035 to 0.0045 in., the moisture content range in Fig. 1 is from 22 pct to approximately 48 pct; at 0.03 to 0.04 in., it is 9 pct to about 22 pct. Any predetermination of moisture content from the relationship presented is useful only for estimating the range or variations that one might anticipate, certainly not for an average that has much significance. This is brought out more emphatically in Fig. 2 by an indication of these moisture ranges along with the corresponding average particle size range as taken from Fig. 1.

This contribution would have added value if the author had included a consideration of other factors affecting moisture content; hence the relationship he presents. He mentions some important factors and refers to "normal operating conditions," which, it is recognized, are different at different plants and often differ considerably in the same plant at different times. It would be helpful if any or all of the following data were available for each of the 26 tests listed for various coal filter cakes: 1—filter area and size of filter; 2—capacity levels at which unit was operated; 3—temperature; 4—solids density of slurry; and 5—vacuum on filter. Pertinent information for Test No. 8 and Nos. 10 through 18 has been reported previously[3, 4] by the author as Plant Nos. 1 to 10 inclusive. Data on other factors such as immersion doubtless merit consideration as he has indicated.

Fuhrmeister[20] has shown the effect of: 1—temperature on both the dryness of calcium carbonate cake and the cake rate in pounds per square foot per hour; 2—percent solids in the feed on the filter cake rate; and 3—percent of —200 mesh solids in a coal slurry of 45 pct solids on both the percent moisture in the cake and the filter cake rate. None of these effects is insignificant. The influence and interrelation of these and/or other factors on cake moisture in coal filtration have been investigated in the laboratory, and a few reports made available;[21-28] no doubt, much other unpublished work exists. In predetermining moisture contents the influence of other factors besides average particle size, as determined by the author, cannot be neglected.

If one plots percent surface moisture against percent of —200 mesh solids in the filter cake, as in Fig. 2, using all the author's data except Test Nos. 19, 22, 23 and 24,** the moisture content spread is no greater,

** These are not included because no —200 mesh is present or it cannot be picked directly from the data.

perhaps even less, than that in the author's plot of surface moisture against average particle size. Thus, within limits, a determination of the —200 mesh in the cake, or slurry, should be all that is necessary to estimate moisture content in the finished cake. Besides, data reported by Fuhrmeister[20] for coal, and other data reported elsewhere[3, 4] by Mr. Lyons, also fit into the general pattern of Fig. 2 as shown. The three points above the upper dashed line in Fig. 2 are suspect because it looks as if the operations were scarcely "normal." One would not select the inside drum and disc combination on the basis of the degree of dewatering accomplished here.† Furthermore, the anomalous positions of the two points for the outside drum above 40 pct moisture suggest that the conditions of the tests represented merit closer scrutiny.

† Comparison of the particle size consist of the coal in Test No. 25 on this filter with that for Test No. 26 on the Genter filter raises a question in regard to the conditions under which Test No. 25 operated. In both these tests the surface moisture reported is virtually equal. In contrast, the —200 mesh material in the Test No. 26 coal is 29.2 pct, or 10 times greater than the 2.8 pct in Test No. 25; 22.7 pct of —325 mesh makes up most of the through 200 mesh in the No. 26 coal, yet the average particle sizes given for the two coals are in the same range.

It is extremely doubtful whether the high proportion of 70x200 mesh, 53.4 pct, in the Test No. 25 coal, which is only 4 times greater than the corresponding fraction, 13.2 pct, 65x200 mesh, in Test No. 26, can account[21-28] for the water-retentive powers necessary in the Test No. 25 fine coal to give surface moisture equivalent to that in Test No. 26. The size characteristics of the No. 25 coal suggest that the surface moisture of the filter cake from the inside drum and disc combination should be appreciably lower than 21 pct.

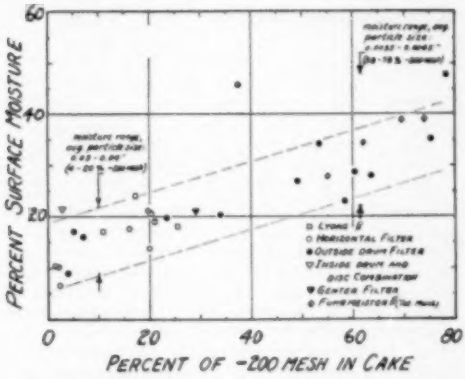

Fig. 2—Moisture range compared with average particle size range shown in Fig. 1.

Comparison of the conditions[5, 6] for these two tests in Table III with conditions in Table IV for the tests by Fuhrmeister shown in Fig. 2 are revealing. One might ask which group represents "normal operating conditions," and why there is such a discrepancy in filter cake rates for the conditions most nearly comparable in the two groups. Are the high-ash clay slimes present to the same extent in Table IV coal slurry as in Table III material, and to what extent do their water-retentive properties affect the latter results?

It would be interesting to compare the filter cake moisture reported for the relatively coarse coals in Test Nos. 22 through 24 with the moisture contents resulting from gravity drainage of the same coals for various lengths of time. In all probability, these could be drained to similar levels of moisture in a car or properly designed pocket within a reasonable time.[21, 22] Certainly, pressure filtering is hardly practical with these coals, and centrifuging would only be justifiable where a premium is placed on time and/or space, or where is shows the greatest overall economy.

It would seem that the author's conclusion relative to the comparative effectiveness of all filters from a moisture reduction standpoint needs to be qualified because the influence of other factors in the relationship he depicts must be included if he is to make the available information most useful. The author's conclusions will require one to exercise the utmost discrimination in predicting dewatering performance when coal slurry solids contain above 30 pct of −200 mesh material, because the test data for three of the four filters, from the author's present paper, shown in Fig. 2 are confined to the left of the 30 pct ordinate. Surely, there is little need for all the types and designs of filters that are available if "apparently all of the makes . . . are equally effective from a moisture reduction standpoint," which, after all, is the purpose of the filtering operation here. There most certainly are significant differences in dewatering ability of filters relative to requirements, conditions, and factors in an operation, else men would never have developed some of the filters now available.[23, 24]

O. R. Lyons (author's reply)—Mr. Clendenin has prepared a very good discussion and has brought up some points that should be clarified.

In the first place, the author has not abandoned the concept that the presence of clays and slimes increases the moisture content of a filter cake. The author merely wished to provide a method that would allow classified filter feeds to be compared with what might be called "normal" filter feeds. The method previously proposed by the author, plotting surface moisture against the product of the −200 mesh pct and the percent ash content of this material, is still valid but cannot be used to evaluate a filter cake containing very little or no −200 mesh material.

Any make of filter will operate at essentially the same vacuum and pull through a comparable volume of air when treating essentially the same size-consist coal. Bed-porosity is related to the size-consist of the material, the type of filter cloth, and the inches of vacuum. Size-consist adequately expresses all of these variables since the type of filter cloth primarily governs the size of the retained material and inches of vacuum depend primarily on the size-consist of the filter cake.

No mention was made of filter capacity in the paper because the author was not able to obtain sufficient operating data to provide more than the rather general conclusion that capacity is related primarily to feed pulp dilution and the size-consist of the feed material. Speed of rotation has an influence on capacity, but it can be neglected, since all filters operate at very slow speeds.

Wide fluctuations in the moisture contents of the filter cakes produced can be explained on the basis of: 1—the size of the filter installation and 2—the nature of the feed, whether classified or not. Large filter installations, where the elimination of blinding and cloth replacement is a continuous rather than an intermittant process, showed very little variation from the average. Small filter installations provided very good or very poor results depending on the degree of maintenance. Semi-classified or classified feeds provide erratic results as compared to "normal" feeds. Channeling can occur in cakes containing only a small amount of slimes; consequently the cake will be wet.

The author cannot agree with Mr. Clendenin's conclusion that there are basic mechanical advantages to be found in one type of filter as compared with another in so far as coal filters are concerned in moisture reduction. It may be true that some filters have decided advantages when compared on the basis of floor space required, ease of maintenance, ease of feeding, and even capacity, but these considerations are outside the scope of the paper.

Table III. Solids in Filter Feed, 50 to 55 Pct

Surface Moisture, Pct	−200 Mesh, Pct	Ash in −200 Mesh, Pct	Filter Cake Rate, Lb Per Sq Ft Per Hr
46.2	37.5	54.8	12
48.3	78.4	48.4	8

Table IV. Solids in Filter Feed, 45 Pct

Moisture in Cake, Pct	−200 Mesh, Pct	Filter Cake Rate, Dry, Lb Per Sq Ft Per Hr
38	74	58
38	69	65
34	62	78
28	55	93

[18] G. B. Dickey and C. L. Bryden: *Theory and Practice of Filtration*, pp. 48, 50. Reinhold Publishing Corp., New York, 1946.
[19] J. W. McBain: *Colloid Science*, pp. 378-379. D. C. Heath and Co., New York, 1950.
[20] Charles Fuhrmeister, Jr.: Improving Filtration and Filtration Rates. *Chemical Engineering Progress* (November 1951) 47, No. 11, pp. 550-554.
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[23] W. T. Turrall: Anthracite Method of Coal Drying. *Coal Mine Modernization Year Book* (1950) pp. 276-294. American Mining Congress.
[24] J. M. Dalla Valle: *Micromeritics*. 2nd ed., p. 283. Pitman Publishing Co.
[25] H. P. Grace: What Type Filter and Why. *Chemical Engineering Progress* (October 1951) 47, No. 10, pp. 502-507.

Comparative Effectiveness of Coal Cleaning Equipment

by O. R. Lyons

DISCUSSION

Judson S. Hubbard *(The Humphreys Investment Co., Denver)*—In this very interesting paper several brief references are made to the Humphreys spiral, a device used for cleaning fine coal.

In Table I, Plants 76 and 77, data are given on spiral performance treating Raton and Trinidad coal. The fine coal, as fed to the spiral in these instances, is actually a table middling, hence the more easily treated material was previously removed and a large number of particles were present which were difficult to clean. Mr. W. M. Bertholf of the Colorado Fuel and Iron Co. presented a paper in February, 1946, *Cleaning Table Middlings from a Coal Washery with the Humphreys Spiral Concentrator,* from which I quote:

"In considering the results of our tests it should be noted that the feed was our table middling, and that any real separation is a 'moral victory' as there is little material that could properly be called coal and practically no heavy rock, the consequences being that previous attempts to clean the middling have not at all been successful."

Referring again to Table I, Plants 72, 73, 74, and 75, these data were obtained by Yancey and Geer and others and presented at the February 1950 Meeting, AIME, in a paper entitled *Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.*

Results shown for those tests involve all particles from 8 mesh through the colloids, which admittedly is not an ideal situation for spiral feed if much refuse is contained in the −80 mesh or −100 mesh size range.

As an illustration of the effect of treating too broad a size range, let us consider Plant No. 75, Kentucky No. 9 seam. Spiral feed was 8 mesh x 0. Now had this been 8x100 mesh the percentage of misplaced material would have been 8.0 pct instead of the reported 15.26 pct. Similar comparisons can be made on the other data presented with respect to the spiral.

Other types of equipment show a similar trend in that whenever too fine a size is treated in a given unit process the percentage of misplaced material increases. Since the spiral is working near the finer end of the size range, it will sometimes be advantageous to treat the entire range of −8 mesh material rather than to deslime and make a fair showing on, say, the +80 or +100 mesh. Desliming is subsequently done in any case in the dewatering or thickening operation.

Results obtained by spiraling any given coal depend on factors too numerous and complex to discuss here, but there are strong indications that proper preparation of feed to the spiral can improve results obtained on some of the raw coals tested. This is clearly pointed out at the end of the aforementioned paper by Yancey and Geer. "The spiral is an extremely simple device which involves no moving parts and is constructed almost entirely of unmachined castings. Since it is such an uncomplicated mechanism, operation is simple and virtually foolproof. These characteristics, which go far toward insuring low cost operation, are attractive attributes in any coal cleaning unit."

Certain equipment used in conjunction with the spiral has resulted in a decrease in the percentage of misplaced material, notably in actual practice the launder screen which is used to remove objectionable high ash fines from a spiral-washed coal product. Private correspondence with the U. S. Bureau of Mines has intimated that an additional yield of coal is possible by flotation of the spiral middling. Possible future improvements and developments may result from other methods now under consideration.

Finally, some compromise must be made between the best metallurgical performance and the best practical or economical results. Mr. Lyons emphasizes in his summary this objective of overall economy in selection of equipment.

G. B. Walker *(American Cyanamid Co., Stamford, Conn.)*—I had the pleasure of reviewing the draft of this paper and my curiosity was aroused by the data given for Tromp plants, in that all of the examples shown appeared to be 2-product separations, whereas all the Tromp plants with which I am familiar have been 3-product units.

The data given for plants No. 101 and 102 appear to be taken from Tromp's brochure on his process and represent the results obtained at the Dominale plant in Holland which was operated for many years by Mr. Tromp. The plant, which was designed to treat 58 tons per hr, was sampled while treating 35 tons per hr of 3¼x¾-in. coal. Example 101 appears to conform to what would result if the middling product were calculated into the refuse product, while Example 102 represents the calculation of the middling into the coal product. It is believed that Examples 103 and 104 represent the operation of the Willem-Sophie Mine in Holland recalculated on the same basis.

In checking the English examples given by numbers 14, 15, 16, and 17, the same procedure seems to have been followed. These results have, apparently, been taken from an article in *Colliery Engineering* in August 1941, describing the initial operation of the Williamthorpe Colliery of the Hardwick Colliery Co. Two vessels are employed in this plant, one to treat soft coal and one to treat hard coal. Example No. 14 presents the results that could be obtained from the soft coal bath if the middling were calculated into the refuse, and Example 15 the results when the middling is calculated into the coal. Examples 16 and 17 represent the same expedient in the case of the hard coal bath.

Of interest to this discussion is the fact that during the past year the Simon-Carves Engineering Co. in England has installed in the Williamthorpe plant their new "Sim-Car" medium cleaning system which is based on magnetic extraction and control and which is licensed under the Heavy-Media Separation Processes patents by the American Zinc, Lead and Smelting Co. This system has been described in the December 1951, issue of *Colliery Engineering.* It is reported that since the Williamthorpe Colliery was changed from the Tromp system of medium cleaning to the Magneto-Motive method of medium control the opera-

Table V. Comparative Effectiveness of Coal Cleaning Methods Employed by the Dutch State Mines

Type of Bath	Media	Type of Coal	Grain Size, In.	Separation, Sp Gr	Washed Coal		Refuse		± 0.1 Distribution, Pct, at Gravity of Separation	Total Misplaced Material in Washed Coal and Refuse in Pct of Raw Coal
					Wt Pct	Sink Pct	Wt Pct	Float Pct		
Shallow	Slime tailings	Bituminous	3½x⅜	1.435	49	0.6	51	0.5	9	0.55
Deep	Barytes	Anthracite	4½x⅜	1.475	76	0.5	24	2.5	30	0.90
Deep	Crushed shale	Anthracite	1x⅜	1.60	75	0.4	25	0.5	4	0.43
Deep	Barytes	Bituminous	1¾x⅜	1.50	86	0.45	14	0.4	15	0.45
Deep	Barytes	Bituminous	1x5/16	1.59	76	0.5	24	1.1	8	0.74

tion of the plant has been greatly simplified in that the viscosity and specific gravity of the medium can be more easily controlled. In addition, we are advised that the medium loss at the Williamthorpe Colliery has been reduced to 40 pct of the figure experienced under the Tromp system. It is our understanding that the National Coal Board is planning on a conversion of the other Tromp plants in England to the "Sim-Car" magnetic recovery and control system.

Example 13 has been identified by Dr. Kriegsman of the Dutch State Mines as being the results obtained in a Tromp washer at the Moritz Mine. This plant was dismantled some 8 years ago and has been replaced by the Dutch State Mines Slate Tailing Process. Dr. Kriegsman has kindly supplied me with the following data, relating to some of the cleaning plants operated by the Dutch State Mines, as a contribution to illustrate the results that can be obtained on Dutch coals with various types of dense medium, including crushed shale, barytes, and slime tailings.

In a survey of this sort it is extremely difficult to cover all phases of the field, and it is my personal feeling that Mr. Lyons has treated the heavy-media system very fairly, although I would like to point out that all his examples are predicated on Link-Belt separatory vessels and appear to be limited to the Champion and Park Breaker installations. While the Link-Belt vessel is one of the best available for use in heavy-media plants, I think it would be only fair to consider some of the other six types of satisfactory vessels which are in operation in the coal industry. There are now 54 heavy-media plants in operation or under construction for the treatment of coal. These 54 plants have a combined capacity of over 25 million tons a year and employ a total of 71 separatory vessels, of which 17 are Link-Belt.

Using Electric Furnaces and Heaters to Determine the Free Swelling Index of Coal

by E. Swartzman and G. C. Behnke

DISCUSSION

W. A. Selvig *(Bureau of Mines, Pittsburgh)*—Considerable difficulty has been experienced by different laboratories in meeting the temperature specifications of the standard A.S.T.M. method, which specifies a gas burner for heating. At the Pittsburgh Station of the Bureau of Mines, where natural gas is used for the test, the gas pressure available is not quite high enough to meet the temperature requirements of the standard method. This difficulty was solved by installing in the gas line a gas booster pump equipped with a diaphragm-valve pressure control. An electric furnace obviously permits easier temperature control than does the standard gas burner.

The degree of swelling of a caking coal, when heated in a crucible, is influenced considerably by the characteristics of the heat input into the coal. In the standard gas-burner method much of the heat is concentrated at the bottom of the crucible, a condition that appears to be duplicated by the conical-style electric heater used by the authors.

The A.S.T.M. standard method for volatile matter in coal specifies platinum crucibles of not less than 10 or more than 20-cc capacity. As the Bureau of Mines has standardized on platinum crucibles of 10-cc capacity, these were used to determine if they would give results for free swelling comparable to those obtained by Swartzman and Behnke, who found 15-cc platinum crucibles to be satisfactory. In these tests the same type of electric furnace and method of procedure recommended by the authors were used. These tests showed that a 10-cc crucible is too small to give results checking those of the standard A.S.T.M. gas-burner method.

The 10-cc crucibles gave buttons that did not conform in shape to the standard profiles, so the free-swelling indexes could not be estimated by matching the shape of the coke buttons with the standard profiles as described in the standard method. It was found, however, that, for coals with relatively low free-swelling indexes of 5 or less, satisfactory agreement with the standard method could be obtained by measuring the profile area or the volume of the coke buttons. For highly swelling coals, satisfactory agreement with the standard method could not be obtained by these methods, as low results were obtained. Apparently, the 10-cc-capacity crucible is too small for satisfactory agreement with the standard gas-burner method for coals having free-swelling indexes greater than 5.

DISCUSSION

Shiou-Chuan Sun (author's correction)—Too late for revision, the author realized that the data on stability of froth presented in this paper were not based on the same height of froth column. It is recommended herein that the stability of froth is better calculated by dividing the time elapsed until the froth disappeared by the volume (in cu cm per sq cm) or height of the froth column, and is expressed in sec per cm. The stability data in Tables I, II, and III of this paper were recalculated, in accordance with this recommendation, and the results are shown respectively in the accompanying Tables VII, VIII, and IX.

Compared with Tables I, II, and III of the paper, the stability data of Tables VII, VIII, and IX are more reliable in disclosing the persistence of froth of various frothers having different frothabilities. For example, the conclusion given in the paper that the stability of froth is generally proportional to the volume of froth becomes invalid when the stability of froth is expressed in sec per cm, as shown in Tables VII, VIII, and IX. Table VII shows that both the stability and volume of froth increase with the addition of small amounts of n-hexyl alcohol, and that further additions of n-hexyl alcohol decrease the stability but increase the volume of froth. Table VIII indicates that the stability of different types of frothers is governed by their chemical composition and is difficult to predict from their frothability. Table IX shows that the stability of different frothers with similar nonpolar groups tends to increase with a decrease of the hydrophilic property of their polar groups, and consequently with the decrease of their solubilities. This is further substantiated by the fact that the stability of frothers, as listed in Table IX, is generally lowered by the solvent effect of absolute ethyl alcohol. Table IX shows also that the stability of froth tends to increase with the decrease of the volume of froth, provided the frothers used have similar nonpolar groups and dissimilar polar groups.

Table VII. Stability of Froth Produced by Various Concentrations of Pure n-Hexyl Alcohol at 7.0 pH Value, and 71.6 CC Per Sq Cm Per Min Rate of Aeration

Test	Conc. of n-Hexyl Alcohol, Mg per L	Volume of Froth, CC per Cm²	Stability of Froth, Sec per Cm	Test	Conc. of n-Hexyl Alcohol, Mg per L	Volume of Froth, CC per Cm²	Stability of Froth, Sec per Cm
1	9.63	1.1	5.36	5	77.04	3.0	3.70
1a	9.63	1.1	5.45	5a	77.04	3.0	3.67
2	19.26	1.3	5.84	6	105.93	3.8	3.68
2a	19.26	1.3	5.77	7	144.45	5.8	2.83
3	28.89	1.7	5.82	7a	144.45	5.8	2.94
3a	28.89	1.7	5.77	8	288.90	6.7	2.69
4	48.15	2.1	4.91	8a	288.90	6.7	2.69

Table VIII. Stability of Froth Produced by Various Frothers, Tested at 7.0 pH Value, 19.2 ± 0.5 Mg Per Liter Concentration, and 71.6 Cc Per Sq Cm Per Min Rate of Aeration

Test	Frother	Stability of Froth Sec per Cm	Stability Index, Using n-Hexyl Alcohol as Standard	Frothability Index, Using n-Hexyl Alcohol as Standard
1	Pure n-hexyl alcohol	5.77	100.0	100.0
2	Tergitol wetting agent 7	<6.00	<104.0	>1153.0
3	Tergitol penetrant 4	6.83	118.2	262.0
4	Ethyl silicate, tetraethyl orthosilicate	3.18	55.2	169.0
5	Methyl Amyl acetate, 4-methyl-pentyl acetate-2	3.39	58.8	138.4
6	Methyl amyl alcohol	4.43	77.0	123.0
7	Pine oil 302, Hercules	5.57	96.5	107.7
8	Cresylic acid 1B, Koppers	2.86	49.6	107.7
9	Cresylic acid No. 1, Reilly	2.92	50.7	100.0
10	Pine oil F, Hercules	5.17	89.7	92.4
11	Cresylic acid D2A	2.42	41.9	92.4
12	DuPont frother B-23	4.83	83.8	92.4
13	Tergitol penetrant 08	7.25	125.7	92.4
14	Butyl carbitol solvent, diethyl glycol monobutyl ether	4.17	72.3	92.4
15	Cresylic acid — English, American Cyanamid	4.08	71.0	84.6
16	Pure n-octyl alcohol	5.27	91.6	84.6
17	Frother 66, American Cyanamid	8.30	144.0	76.9
18	Pine oil GNS, No. 5, American Cyanamid	4.50	78.1	76.9
19	Cresylic acid EE, Oronite	4.42	76.7	73.1
20	Cresylic acid 4020, American Cyanamid	8.22	142.6	69.2
21	Pine oil 302W, Hercules	4.88	84.8	69.2
22	Frother HTT, American Cyanamid	5.00	86.8	61.5

Table IX. Stability of Froth Produced by the Principal Constituents of Pine Oil, Tested at 7.0 pH Value, 19.2 ± 0.5 Mg Per Liter Concentration, and 71.6 Per Sq Cm Per Min Rate of Aeration.

Principal Constituents of Pine Oil	Polar Group	Solubility, Gm per L	Stability of Froth Sec per Cm	Stability** Index, Using n-Hexyl Alcohol as Standard	Frothability Index, Using n-Hexyl Alcohol as Standard
Beta terpineol	C-OH	2.2	3.85 (3.31)*	66.7	100.0
Alpha terpineol	C-OH	1.98	4.36 (4.00)	75.7	94.6
Fenchyl alcohol	C-OH	sl.	3.54 (2.89)	61.4	100.0
Borneol	C-OH	0.64	3.45 (1.78)	59.9	84.6
Camphor	C=O	1.6	3.36 (3.84)	58.3	84.6
Dihydro-terpineol	C-OH	—	4.57 (3.22)	79.3	53.7
Fenchone	C=O	i	7.25 (3.40)	125.5	30.8
Anethole	C-O-C	v.sl.	8.33 (3.36)	144.2	23.3
Methyl chavicol, estragol	C-O-C	i	10.0 (4.33)	173.5	15.4
Terpinolene	none	i	(3.25)		tr.
Dipentene	none	i	(4.17)		tr.
Camphene	none	i	(2.89)		tr.
Beta pinene	none	i	(3.13)		tr.

* Data in parentheses were determined by first dissolving reagent in equal volume of absolute ethyl alcohol.
** Stability of froth of n-hexyl alcohol used for calculation was 5.77 sec per cm.

Geology of the Silver-Lead-Zinc Deposits of the Avalos-Providencia District of Mexico

by W. H. Triplett

DISCUSSION

John G. Barry *(Mexico)*—"The Nerinea limestone," writes W. H. Triplett, p. 584, "is . . . medium-bedded (0.5 to 1.0 m). . . ." This is the Zuloaga formation. It has always been described as thick-bedded (1.0 to 3.0 m). "Whether the marls and shales were less amenable to replacement," he goes on to say, ". . . is a matter of opinion." In general, it is probable that the thinner-bedded formations were less competent than the near-by thicker-bedded, so that the latter produced pre-mineral breccias when all were subjected to flexing stresses.

With reference to Mr. Triplett's analysis, p. 584, under *Lower Cretaceous*, in the Taraises and Cupido formations some few horizons should be considered as possibly containing hidden orebodies. In the Potrero camp, shown on the general map in the northwestern part of Fig. 4, p. 587, several mineralized beds are known; they are found throughout the whole thickness of the Cuesta del Cura formation.

With regard to intrusive rocks, p. 585, the general map, Fig. 4, shows that the Potrero and Noche Buena camps are near a third, monzonitic intrusive. It seems doubtful that the Aranzazú was any hotter than the Providencia camp. However, the former does contain large deposits of exogenetic silicates, which in large part occur within the Zuloaga limestone, as shown by Bergeat" in a stereogram. At the Aranzazú camp the La Caja formation was only slightly changed, whereas it was altered highly at the Providencia, possibly endogenetically. That the temperatures at the Providencia camp were relatively high at one time is also shown by the tourmaline veinlets and lenses in the monzonite and by the garnets on Temeroso peak; high temperature mineralization has possibly been eroded.

Maps and sections show that the Concepción del Oro —Temeroso stock occupies the original locus of some 40 km³ of sedimentary rocks. The remains of large roof pendants at Carmen and Animas indicate that stoping was active. Whether all the sedimentaries were stoped and assimilated presents a lively problem, a correlative of which would be an estimate of the original constitution of the invading instrusive rock.

As to extrusive rocks, see p. 585, there is also a large thick area of the extrusives left near Noche Buena in the "La Caja" intramontane valley of the La Caja range. This place gave its name to the range. The volcanics are also said to occur in considerable amount in the Mazapil valley."

Turning now to the remarks under Regional Structure, p. 586, as partly indicated on the general map, Fig. 4, the Mazapil Mountains of the Mazapil region are a continuous, U-shaped range including a northern leg of some 24 km, a southern leg of similar length, and a uniting base which lies to the East and which measures some 10 km in a S 20 E direction. Thus the U-shaped mountains are continuous through a length of 58 km and also contain one continuous anticline, the adjacent valleys being synclinal. The northern leg of the U is called the La Caja range, the base is called the Concepción del Oro range, and the southern leg the Santa Rosa range.

The ranges in northeastern Zacatecas and southern Coahuila contain anticlines and the valleys contain synclines, which where thin are necessarily highly compressed isoclines with very steep dips. Many of the anticlines also have isoclinal components which are overturned, original mushroom tops being indicated. The anticline in the base of the U of the Mazapil Region, Concepción del Oro range, is a recumbent anticline, over to the ENE, with a necessarily subjacent recumbent syncline.

Table IV. Recast of Table II

		Assays				Contents, Pct				Contents, Pct			
						Gold		Silver		Lead		Zinc	
Level	Tons	Au, G	Ag, G	Pb, Pct	Zn, Pct	Diff.	Cum.	Diff.	Cum.	Diff.	Cum.	Diff.	Cum.
11-12	10,463	0.82	339	11.0	20.8	5.7	100.	3.8	100.	3.8	100.	2.4	100.
12-13	60,496	0.44	240	8.2	22.1	17.6	94.3	15.6	96.8	16.6	96.2	14.6	97.6
13-14	91,431	0.31	199	6.2	20.7	18.5	76.7	18.8	80.8	18.8	79.6	20.9	83.0
14-15	88,422	0.31	201.5	6.0	19.4	18.5	58.2	20.4	61.8	18.3	60.8	19.2	62.1
15-16	80,343	0.31	195.5	6.6	17.9	16.4	39.7	16.4	41.4	17.8	42.5	15.6	42.9
16-17	71,993	0.29	185	6.0	18.6	13.3	23.3	13.7	25.0	13.7	24.7	14.1	27.3
17-18	41,522	0.25	160	5.3	17.6		10.0		11.3		11.0		13.2
18-19	28,157	0.21	160.1	4.8	19.4								

Table V. Recast of Table III

		Assays				Contents, Pct				Contents, Pct			
						Gold		Silver		Lead		Zinc	
Level	Tons	Au, G	Ag, G	Pb, Pct	Zn, Pct	Diff.	Cum.	Diff.	Cum.	Diff.	Cum.	Diff.	Cum.
7-8	9,915	1.33	381.1	14.7	18.5	2.7	100.0	1.6	100.0	1.5	100.	1.2	100.
8-9	39,855	1.11	454.6	14.3	19.5	9.3	97.3	7.6	98.4	5.9	98.5	5.2	98.8
9-10	84,080	0.86	335.6	13.3	17.9	14.4	88.0	12.1	90.8	11.2	92.6	9.9	93.6
10-11	144,125	0.69	310.5	13.7	17.1	20.4	73.6	18.3	78.7	20.1	81.4	16.6	83.7
11-12	189,781	0.45	265.8	12.4	16.9	10.3	53.2	12.2	60.4	14.1	61.3	12.7	67.1
12-13	96,744	0.44	210.1	9.4	16.9	8.8	42.9	8.4	48.2	9.3	47.2	11.0	54.4
13-14	110,544	0.42	216.3	9.9	16.1	9.5	34.1	9.8	39.8	11.3	37.9	12.0	43.4
14-15	152,110	0.52	241.1	9.2	16.5	16.0	24.6	14.8	30.0	14.0	26.7	16.6	31.4
15-16	131,430	0.38	338.6	11.8	19.3		8.6		15.2		12.7		14.8
16-17	5,927	0.27	184.3	4.6	19.9								

The map prepared by Dill, Fig. 5, is similar to one prepared by the writer in 1950, King's map having been used as a base in both cases. It will be seen that a prominent axis of horizontal flexure of the anticlines in the "kingdom" will pass through the Mazapil region, which thus becomes a prominent structural focus in an extremely large area of the Mexican geosyncline.

The Potrero and Noche Buena Camps occur in the western part of the La Caja range. In general, the Providencia Camp is near the curve of the northern leg into the base of the U. The Concepción del Oro Camp is in the base of the U. The La Laja Camp is at the swing of the base of the U into the southern leg, while the Santa Rosa Camp is west of the last and in the Santa Rosa range.

The Santa Rosa rhyolites to which Mr. Triplett refers, p. 586, were called by Burckhardt, following determinations by H. Rosenbusch, a highly altered porphyry which was not exactly determinable. The writer believes that is may represent a chilled facies of the monzonitic rocks.

The vertical beds referred to under Local Structure, p. 586, are necessarily seen in passing from the recumbent anticline to the subjacent recumbent syncline.

The occurrences at Noche Buena and San Francisco del Alto are another structural story, but the latter place is centrally located in the La Caja range and not far from a central synclinal flexure in the anticline.

"Shrinkage of the intrusives," see pp. 586-587, probably did not cause openings in the limestone beds; rather these were due to brecciation caused by small horizontal flexures. It seems likely to the writer that in the deeper levels the flexures broke to form the fractures, or veins, mentioned.

If on the maps of Fig. 7 a series of lines are drawn parallel to the sedimentary markers and contacts, it will be seen that many widely separated bodies are aligned. This indicates deposition in conjunction with favorable beds. The writer knows of several identifiable favorable beds in the Zuloaga limestone.

The writer believes that the facts presented under Zoning, p. 591, are a contribution of very great importance from all points of view, and he further believes that much credit should be given to Manager E. A. Manderfield for his constant and careful statistical work during the past 28 years. This collection is extraordinary, and Peñoles should be congratulated. Furthermore, it will be desirable to collect similar facts from many large mines, but with additional details such as the variations in contents of pyrite and other minerals.

For the two best-known and deepest orebodies the facts are noted in Tables II and III, which show tonnages between equally spaced levels, and the corresponding assays and metal ratios. These data have been drawn in rectilinear graphs. From the viewpoint of genesis the writer believes that it might be helpful to calculate the total contents of each metal and its vertical distribution from the bottom up.

When plotted on rectilinear charts, Tables IV and V show almost straight and parallel lines; other methods of plotting would possibly be better. The greatest spread between gold and zinc is in the case of Table V, between Levels 13 and 10, lead and silver lying between them.

Table IV shows that the increments of gold are less than the others up to Level 13, and greater above. The increments of silver and lead are closely related, and they are relatively large between Levels 16 and 13. The increments of zinc are larger than the others up to Level 16, and between Levels 15 and 13.

Table V shows relatively large increments of gold up to Level 14, and between Levels 12 and 9. The increments of silver and lead are closely related, but they show relatively large increases between Levels 16 and 14, and between Levels 12 and 9. Increments of zinc materially exceed others up to Level 12, above which they are not much less than those of silver and lead.

The writer concurs with the author in the belief that the collection and study of similar data in many other cases will in the future be a great aid to an understanding of ore deposition and stabilities, and to predictions resulting therefrom.

The general map, Fig. 4, is similar to a part of one prepared by the writer in February 1950, which was widely circulated among mining persons and companies in the Mazapil region. One important correction should be noted. There is no lobe of intrusive at Potrero, and the eastern point of the intrusive does not extend so far to the east of Noche Buena. The northern and eastern parts of the boundary of the San Francisco del Alto-Noche Buena monzonitic intrusive should run as follows: from 9980 N-14,000 W to 9975 N-12,700 W, and thence southerly. The Cuesta del Cura-Indidura contact is at the Potrero Adit (19), the latter formation lying to the north.

It is important to emphasize that the sedimentary column compiled by the present writer, see Fig. 1, is still subject to use and checking lithologically. C. L. Rogers, U. S. Geological Survey, and his associates from Recursos Minerales who are studying phosphate occurrences are using a column about as follows:

Cretaceous	Caracol	Plus	1000 M	
	Indidura		180	(two members)
	Cuesta del Cura		270	
	La Peña		140	(two members)
	Cupido		300	
	Taraises		140	
Upper Jurassic	La Caja	Avg	100	
	Zuloaga	Plus	400	(six units)

It is even more important to note that the hiatus at the top of the upper Cretaceous is one of extensive time. It represents the Laramide Revolution during which the sediments were uplifted and lithified, and compressed and folded intensely. The uplifting permitted the fusion and welling up of those intrusive rocks which are now seen as monzonites. Erosion was probably continuous and intense. The Laramide Revolution probably began in the late upper Cretaceous and continued into the Eocene. It is interesting also to take into account some approximate secular times which have been reported as measured by radioactivity:

Appalachian Revolution	210,000,000 B.C.
End of Triassic	150,000,000
End of Cretaceous	60,000,000
Laramide Revolution	
Miocene	10,000,000

Successive disturbances of equilibrium produced openings through which flowed differentiates from the magmatic depths; from them successive stages of mineralizations were deposited. The writer believes that there were *zones* which developed mechanically outward from certain centers of resistance and reinforcement. He would also prefer to think of differences found in fairly uniform deposits, such as those at Providencia, as *segregations*.

Much careful work on collections and laboratory studies remains to be done before the several stages of mineralizations can be competently discussed. However, at the present time the writer believes that there were some seven stages of mineral aggregates deposited during the Laramide Revolution and that the principal stage found at Providencia was mesothermal and one of the last.

The much later andesitic and rhyolitic rocks of the Miocene, dikes at San Pedro de Ocampo as well as the volcanics noted, were followed by mineralizations which deposited siliceous-silver, siliceous-gold, and calcitic-gold ores. Still later disturbances in the Tertiary were probably responsible for the sparse and scattered deposition of minerals related to manganese, fluorite, mercury, antimony, barite, and various carbonates.

[9] Alfred Bergeat: La Granodiorita de Concepcion del Oro en el Estado de Zacatecas y sus Formaciones de Contacto. Bol. Inst. Geol. de Mexico (1910) No. 27, p. 109.

[10] Personal communication from C. L. Rogers, U.S. Geol. Survey.

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aime NEWS

Industrial Minerals Div. Formulates L. A. Program

The Industrial Minerals div. has been forging ahead in formulating a program for the February Annual Meeting in Los Angeles. Several papers have been listed and more are being worked into the program.

For the cement, lime and gypsum part of the sessions, papers on *Gypsum in the Aircraft Industry, Geology of San Marcos Island Gypsum Deposit, and Status of New Dry Rotary Blast Hole Drills*, are expected.

The Chemical Raw Materials sessions will feature a symposium on potash, with 13 papers as the anticipated number to be presented. Airborne and Ground Magnetometer Work has been suggested. Eight papers have been set for Mining Geology, Four papers, *Mining Phosphate Rock, Idaho Phosphorous*, and *Geology of Carlsbad Potash* and *Geology of Pearlite* will be on the program.

Sessions on Dimension stone and slate have been tentatively worked out with papers on *Geologic Factors Affecting the Dimension Stone Industry of California, A Quarter Century of Granite Quarrying in Massachusetts, Rock Bursting and Other Geologic Problems in the Granite Quarries of Barre, Vt., Quarry Practices in the Indiana Building Stone Industry, Stratigraphy and Composition of the Salem Limestone, and Dimension Stone in the U. S. in 1952.*

The Rare Minerals Committee is aiming for six papers with two papers now definitely on the program.

Junior Application Procedure Changed

T. B. Counselman, Chairman of the Membership Committee, requested the Board to reconsider its action on Feb. 17, 1952 requiring that the signature of the original member sponsor be reobtained on the application for Junior Membership plus one more member's signature. The Board discussed this matter and voted that hereafter, a Student Associate applying for Junior Membership shall obtain only one member's signature, such signature to be other than that on his application for Student Associate. However, if he does not transfer in the year of his graduation, he must obtain the usual three members' signatures.

One of the stops on the Los Angeles Specials will be Kennecott Copper's Bingham Canyon mine. Delegates will also visit Hoover Dam, during the stopover at Salt Lake City.

All Aboard AIME Special For LA Annual Meeting Stopovers at Salt Lake City and Las Vegas

All Aboard! All Aboard the AIME Special leaving from Chicago, Friday, February 13 for the Annual Meeting in Los Angeles. The train will arrive in Los Angeles February 16, after stops at Salt Lake City and Las Vegas, Nev., for sightseeing trips to Utah copper mines and Hoover Dam.

Highlighting the stopover at Salt Lake City will be a visit to the open pit in Bingham Canyon, with a stop at one of the most famous western mining towns, Bingham. Transportation will be by Grayline Sightseeing buses. The train arrives at Salt Lake City at 4 pm Saturday.

At Las Vegas, a bus tour to Boulder City and Hoover Dam has been planned with a sightseeing jaunt to Las Vegas' famous casinos following. Lunch and dinner will be in the downtown area of Las Vegas. Arrival at Las Vegas is 9 am Sunday.

Roundtrip first class railroad fare from Chicago is $145.42, including Federal tax. Lower standard berth, one way is $26.80. A roomette, one way, for one, will cost $37.49, while a bedroom for two is $58.94. A compartment for two, one way, is $74.87.

Tour privileges at Salt Lake City are $3.40 for the three-hr side trip, and $1.27 at Las Vegas.

Reservations should be sent to the Union Pacific Railroad, 1 South LaSalle St., Chicago 3, in care of John C. Pollock, Traveling Passenger Agent.

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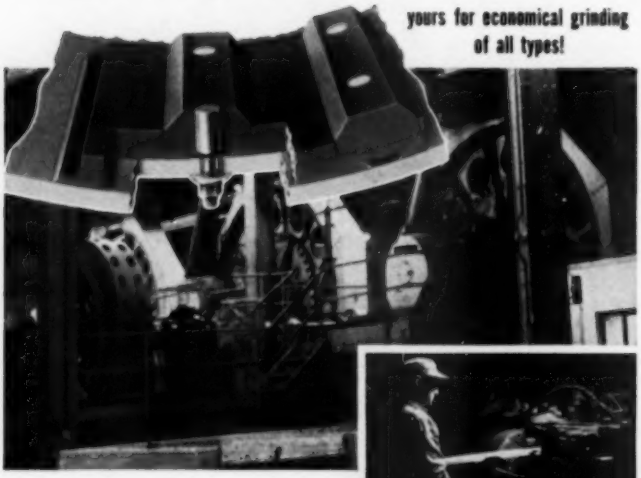

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Seek Recommendation For Kelvin Medal Award

Secretary Robie has received a letter from the Secretary of the Institution of Civil Engineers in London, England, requesting from each of the leading engineering societies in all parts of the world the name of a person recommended for the award of the Kelvin Medal in 1953. This medal, which was awarded for the first time in 1920, is presented triennially as a mark of distinction in engineering work or investigation of the kinds with which Lord Kelvin was especially identified. Although Lord Kelvin was pre-eminently a pioneer in the study of pure physics, he also achieved a great deal in the application of pure science to the solution of engineering problems. It is stressed, therefore, that the Medal be awarded to men of distinguished service in the application of science to engineering rather than in the development of physical science itself. The AIME has been requested to send a recommendation for this award before Feb. 1, 1953. A short statement of the grounds upon which the recommendation is made should accompany the nominations. Secretary Robie will be glad to consider any recommendations from AIME members for presenting to the Board for final selection.

Earthquakes Discussed

The recent California earthquakes were discussed by George Shor, seismologist, and member of the AIME, at the last meeting of the Black Hills Section, held at Lead, South Dakota.

The business end of the meeting involved distribution of membership drive material.

Lima-Peru Section Holds Lunch Meeting

The second luncheon meeting of the Lima-Peru Local Section of the AIME was held at the Golden Room of the Hotel Bolivar with John R. Neale, Director of the Institute of Inter-American Affairs acting as guest speaker.

His talk was entitled *A Discussion of the Point Four Program in Peru.* The Lima-Peru Section is one of the most recent added to the AIME. Its territory includes Lima and Callao and their suburbs.

Utah Section Meets At Salt Lake City

Stanley Hughes, assistant to the general manager, Western Operations, Kennecott Copper Corp., presented a paper on the *Kennecott Utah Copper Refinery* at the Utah Section meeting.

The meeting was held at the Newhouse Hotel, Salt Lake City, with W. G. Rouillard, as chairman.

UPADI Meeting at New Orleans Attracts Engineers of 14 Nations

The second meeting of the Pan American Union of Engineering Societies (UPADI), held at New Orleans during August, was attended by a total of 315 persons representing 14 nations.

UPADI held its first meeting in Havana, April 1951 with 59 delegates from 19 nations attending. The aims of UPADI, as stated at the first meeting, are to encourage, promote, expand, and guide the work of engineers in the Americas. Plans were made for periodic Pan-American engineering congresses and expositions that will promote individual and collective visits by member countries.

Engineers Joint Council and Engineering Institute of Canada were co-hosts at the Second Convention. The local organizing committee was under the direction of James M. Todd, who was also chairman of the U. S. delegation. Facilities were provided by Tulane University.

Present Plaque

Highlighting the convention was the presentation of a plaque commemorating the Second UPADI Convention to the Engineers Joint Council. Carlton Proctor accepted the plaque on behalf of EJC.

Officers of UPADI extended an invitation to all engineers in the Western Hemisphere to take part in the organization's activities, particularly those working in or interested in Latin America.

A series of technical papers were presented at the meeting. They dealt with engineering education and included authors from Argentina, Brazil, Cuba, Dominican Republic, Peru, Uruguay, and the United States. Papers were also given on economic development and financing in Latin America. Social events took place at International House, and the Colonial Country Club.

The primary purpose of the convention was the adoption of a constitution and by-laws. Copies were prepared in English and Spanish and distributed during the meeting. Also made part of the official record was a Code of Ethics developed by the Committee on Professional Ethics and adopted in the resolutions of the convention. Another part of the agenda was the budget for the coming year. The convention fixed the amount at $15,000. Committees on administration, professional ethics, relations with other organizations, and technical matters were established during the meeting.

The delegates took a number of side trips which included the Kaiser Aluminum Works, the Port of New Orleans, the U. S. Experimental Waterways Station at Vicksburg, Miss., and the Freeport Sulphur Co.

The U. S. delegation to the UPADI convention at New Orleans included: Engineers Ackerman, Agthe, Browne, Hathaway, Proctor, Pratt, Pope, Todd, Burt, Kelly, Tyler, Lederer, Carey, Reimel, and Steinberg.

U. S. Participation

Generally considered to be a success, some made the comment that stronger U. S. participation is needed if UPADI is to live up to its potential as a force for the betterment of international relations between engineers of the U. S., Canada and Latin America.

Best representations came from Argentina, Brazil, Canada, Chile, Colombia, Cuba, the Dominican Republic, Mexico, Panama, Paraguay, Peru, Uruguay, and the U. S. No delegates from Bolivia, Costa Rica, Ecuador, Salvador, Guatemala, Honduras, Haiti, Nicaragua, nor Venezuela were present.

Stacy May, economist with the International Basic Economy Corp., in an address to delegates pointed out that the United Nations index for Latin American minerals production indicates that the area is far behind in development. Overall production increase for the period 1937 to 1947 was 33 pct, only 3 pct greater than population increase. Most of the growth was represented by an 88 pct increase in petroleum production, chiefly in Venezuela. Other figures indicated a mixed trend. While bauxite, iron ore and zinc production showed tremendous increases, "copper, tin, manganese, and lead from Latin America showed no long term growth trend."

He said, "in metals particularly, there is strong evidence that the world is in a position far more serious than an emergency shortage. If this be true, and I believe it is, it should provide a condition favorable to Latin American developments in this field upon a very considerable scale."

Foreign Investment

May said that industrial development might be financed from domestic savings to the extent of about $2 billion, but at least $500 million must come from foreign sources.

The Convention elected Carlton S. Proctor, president of ASCE, as its president. Other officers were Manuel J. Puente, Cuba, first vice president; Saturnino de Brito, filho, Brazil, second vice president; S. E. Reimel, U. S.; Rodolpho Ballester, Argentina; Juan V. Cabrerizo, Peru; Hector Cassaigne, Mexico; J. B. Stirling, Canada, secretaries. Luis Giannattasio of Uruguay was elected honorary president by acclamation.

The convention will hold its next meeting in 1954 at Sao Paulo, Brazil, when that city celebrates the 400th year of its founding.

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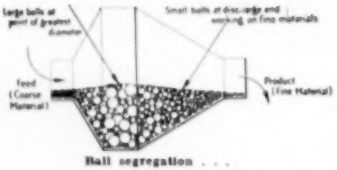
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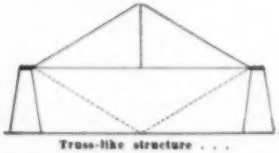

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Fifth Empire Mining Congress in Australia

The Fifth Empire Mining and Metallurgical Congress, scheduled for Australia, in April and May of 1953 will include every major mining field and metallurgical plant on the continent in its list of tours.

The plenary session will be held in Melbourne beginning April 20, but before it gets underway, alternative tours have been arranged in Tasmania, each occupying one week.

During the week the Congress opens, visits have been arranged to various parts of Victoria. Canberra will be visited April 26, and members will be the guests of the Commonwealth Government. From Canberra, members will go to Sydney, where three alternative tours have been arranged.

Following the conclusion of the Australian section of the Congress on May 22, delegates may travel to New Zealand by air. Two tours have been arranged. One, of five days, is largely recreational. The other, 16 days, will cover gold dredging, coal mining, hydro-electric power plants, and geothermal steam experiments.

St. Louis Section Meets

Fred J. Meek was elected section delegate and Clyde Smith named alternate at a recent meeting of the St. Louis Section at the Hotel York.

Charles R. Gebel, Kaiser Aluminum & Chemical Sales, Inc., presented a film, *Bricks from the Sea* and discussed refractories. The group voted 13 to 5 to adopt the title *American Institute of Mineral Engineers*. Sixteen members abstained. The mail ballot showed an 11 to 1 majority in favor of the same title.

Initiation Fee Aids Local Section

The benefit to Local Sections in obtaining new members is not only an increase in membership but also financial aid. It may not be generally known that a Local Section receives $2 of the initiation fee of each new Member or Associate Member who resides in the territory of its Section. This does not apply to change of status nor to reinstatements. Until recently, only 10 pct of any installment paid on the initiation fee was sent to a section but now the full $2 is sent at once although the full initiation fee may be paid, if the member so wishes, in four equal installments. This addition to a section's working capital should be an extra incentive toward obtaining new members. Checks are mailed to Local Sections at approximately quarterly intervals.

Nevada and California Papers at S. F. Section

The San Francisco Section met at the San Francisco Engineers Club to hear D. F. Hewett, staff geologist for the U. S. Geological Survey, Pasadena, Calif.

He spoke on the discovery and development of the rare earth deposit at Mountain Pass, San Bernardino County, Calif. Using color slides, Hewett described the geology of the discovery in detail.

A. E. Miller, general manager of the Yerington mine, wrote and A. J. Gould presented a paper, *Anaconda's Activities at Yerington*, at the opening meeting of the fall series.

Dues Bylaws Revised

In view of the authority given by the Board by referendum of the membership, at its meeting in Chicago on September 7, the Board voted to revise the first two sentences of Article II, Sec. 2 of the Bylaws to read as follows:

"Until otherwise determined by the Board of Directors of the Institute, the dues of Members and Associate Members shall be $20 per annum, and dues of Junior Members shall be $12 per annum for the first six years of membership and thereafter $17. Dues shall not be required of Honorary or Senior Members."

Northwestern Section Celebrates Centennial

AIME members in the northwest contributed a large share to the celebration of a Centennial of Engineering in Spokane, Wash. Much of the program was directed at the schools.

Dwight Myers, local section secretary, was in charge of publicity, while R. O. Oscarson and R. D. Brown, of Allis-Chalmers, and S. E. Maddigan and P. P. Zeigler of Kaiser Aluminum helped to present the program to 29 schools. F. R. Morral represented the AIME in doing a good share of the school program preparation and planning. He also assisted in preparing the rest of the program, including presentations to the various national service organizations. R. B. Mears spoke to the Chamber of Commerce.

Carlsbad Group Hears Douglass on Europe

The Carlsbad Potash Section held a meeting at the Riverside Country Club, Carlsbad, N. Mex., with G. C. Weaver presiding. E. W. Douglass, Potash Co. of America spoke on the *European Potash Industry*.

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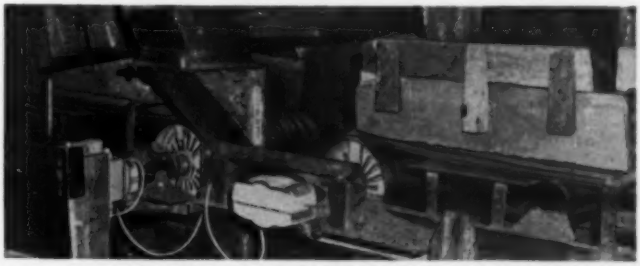

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THE DRIFT OF THINGS

by Edward H. Robie

Round Trip to Spokane

SEPTEMBER usually being a fine month for motoring, we set out with our better half at the end of August in our Studebaker for points West. A combined business trip and vacation. The first night found us 382 miles away, at the Statler Hotel in Buffalo, and the next morning we revisited Niagara Falls for the first time in some forty years. The cataracts were as impressive as ever and certainly worth a day of any vacationist's time, instead of the half hour we could spare before going on to Chicago.

The Centennial of Engineering, the hundredth anniversary of the founding of the ASCE, no doubt attracted the greatest gathering of engineers in history but still the attendance was disappointing to most of the forty or fifty societies that participated. Perhaps the time—the two weeks following Labor Day—was unfavorable. Few could afford to spend ten days away from home and business at that time of the year. Fall meetings of the AIME normally attract far more than the 500 that registered in Chicago. The technical program, however, was excellent, especially that arranged by the Minerals Beneficiation and Industrial Minerals Divs. The mineral industry papers of a more general nature that were presented as a part of the Centennial program on Monday and Tuesday of the following week were of high quality and offered by top names but had a meager audience.

Undoubtedly the celebration brought engineers and engineering to the attention of millions of people. The pageant at the Museum of Science and Industry proved continuously interesting, showing the progress of engineering following the first use of fire. The show continued for many weeks. A special stamp was issued by the Government, which engineers might well use on their Christmas cards or other mail this year.

After six days in Chicago we set out north, first visiting Fairbanks Morse in Beloit and Allis-Chalmers in Milwaukee. Will Mitchell's Basic Industries Research Laboratory is doing interesting work and being installed in more comfortable quarters. The next day at Ishpeming, C. W. Allen arranged for luncheon with a local group of AIME members and we then looked over the new surface plant of the Mather B mine, a model of modern functional design. Then on to a small dinner meeting at Calumet following cocktails at Endicott Lovell's attractive summer home on the lake at Point Mills. The new Memorial Union Bldg. at Houghton, where we were domiciled for the night through the courtesy of Ray Drier, is a beautiful modern addition to the college, a model of its kind.

Lunch the next day in Duluth with a school-day friend, Walter Mesaros, who is directing Oliver Iron's taconite research, and then on to Virginia for an evening meeting of the Minnesota Section's Minerals Beneficiation group. An attendance of 65 testified to the success of the Minnesota Section's plan of bringing the AIME to local members on a divisional basis. A mining group is similarly active. Each holds about four meetings a year, these supplementing the annual meeting of the Section in mid-January in Duluth. The next morning was spent in recruiting some of the big iron guts on the Mesabi Range with Bill Taylor and C. J. Calvin, and then on to Grand Forks, N. D.

Grand Forks has a fine new Bureau of Mines building, to be devoted to fuel technology, and of course the University of North Dakota. A mining curriculum is well established, as is an AIME Student Chapter. Such well-known MBD'ers as Grover Holt and Joe Swainson graduated there. By nightfall the next day we were at Williston, where who should we run into but Charley and Eda Milliken, who the next morning showed us over Amerada's wells in the new and famous Williston basin thereby considerably increasing our knowledge of petroleum production. That afternoon we saw the Fort Peck dam, the biggest earth fill dam in the world, some four miles long and about a mile wide at one point. Later we saw the Hungry Horse dam in Idaho under construction, the fourth biggest concrete dam in the world. In between we drove through Glacier Park on the famed Going-to-the-Sun Highway, perhaps as beautiful a mountain ride as there is in the country.

High spots of the next week started with a luncheon meeting with Columbia Section officers at Spokane. Incidentally, Spokane has a beautiful new modern hotel, the Ridpath, which we can recommend. Then on to Wallace for a dinner meeting with some of the Coeur d'Alene Subsection members. The next morning, a trip around the district and a visit to ASARCO's new Galena mine. Lunch at the attractive new Florence Hotel at Missoula, and a never-to-be forgotten dinner with Chester Steele and other Montana Section notables that night at Teddy Trepartiach's Rocky Mountain Cafe at Meaderville. No gambling left there but plenty of good food and drink. A little sightseeing the next morning on "the richest hill on earth" and in the afternoon into Yellowstone Park. A page or two could be devoted here to the pitfalls of visiting national parks after the season has closed.

The next day a trip of over 400 miles through Wyoming, winding up with a dinner meeting with a Wyoming Section group at Laramie, at the Diamond Horseshoe no less. And Saturday night down at Denver a fine meeting of the Colorado Section.

Then six days at the Mining Congress meeting, perhaps the biggest aggregation of mining men ever held, with a fine machinery show. It started Sunday afternoon with a trip to Golden to see all the new buildings and equipment at Colorado's world-famous mining school; and wound up, for us, on Friday with a trip to the famous mining district of a half century ago —Cripple Creek. This included a mountain trout luncheon at the Elks Club in Victor, underground in the Ajax mine, and a steak dinner at the famous Broadmoor in Colorado Springs. There's nothing like a mining field trip to establish friendships which often prove lasting.

Denver to Chicago took the better part of three days. Here the trip was punctuated by a flight down to Houston for the Petroleum Branch fall meeting. More than 1500 men and close to 600 ladies registered, the Branch's biggest meeting ever. An earnest, young, live and friendly group who have contributed a great deal to the progress of the Institute in the last few years. A fine buffet supper and water show at Glenn McCarthy's famous Shamrock.

And finally home. No accidents of any kind, nor did we even see one. Practically perfect weather, warm and only a couple hours of rain in five weeks. Some 6700 miles covered, at 20.7 miles per gal actual mileage, 1 gal less than the speedometer reading, most speedometers including ours, being fast. Western highways excellent, and faster than in the East, but the Pennsylvania Turnpike, now 355 miles long, the finest of all. A memorable trip. We are more than ever proud of our profession and our country.

JOHN M. DAVIS

John M. Davis has left Western Machinery Co. to take a job as project engineer with Baroid Sales Div., National Lead Co., and is now located at Houston, Texas.

C. W. Allen, general manager of the mining dept. of the Cleveland Cliffs Iron Co., visited the firm's iron mines on the Mesabi iron range of Minnesota recently.

Thomas P. Anderson, mining engineer and geologist, has resigned from the Raw Materials Div. of the U. S. Atomic Energy Commission where he was assistant chief of the Denver Exploration Branch. He will enter private practice in Golden, Colo.

James E. Atkinson, consulting mining engineer and geologist, left New York for Colombia, S. A., as general manager for Cia Minera Choco-Pacifico, S. A., operating five dredges in gold and platinum on the Pacific coast of Colombia.

Russell H. Bennett has been elected a director of Placer Development, Ltd. with head office in Vancouver, B. C., and is also a director of its subsidiary, Canadian Exploration, Ltd., which mines and mills lead, zinc, and tungsten ores at a property near Salmo, B. C.

Max M. DuBois, general superintendent of the U. S. Smelting, Refining & Mining Co.'s U. S. and Lark mine, has become assistant to manager of western mines, succeeding **J. M. Ehrhorn** who is being assigned to special work in the company's western operations. **Benton Boyd,** superintendent of the Lark section, succeeds DuBois as general superintendent. **Harold Wells,** assistant superintendent of the Lark Section, is promoted to superintendent of this section. **Clyde Gillman,** engineer, succeeds Mr. Wells as assistant superintendent at Lark.

John F. Haynes is mining engineer with the Utah-Wyoming Branch, Mining Div., Region IV, U. S. Bureau of Mines.

Dale E. Hirschfeld has been accepted as an engineer in training by the Jeffrey Mfg. Co. in Columbus.

Allen B. Hollett has resigned as assistant chief engineer with Cleveland-Cliffs Iron Co. on the Mesabi range to join AS&R's mining department in New York City.

Ralph C. Homer, formerly teaching at Colorado School of Mines, has joined the exploration staff of Kennecott Copper Corp., New York City as chief geophysicist.

James R. Johnson has returned to Libby, Mont., as mine engineer with the Zonolite Co., after 14 months in the U. S. Marine Corps.

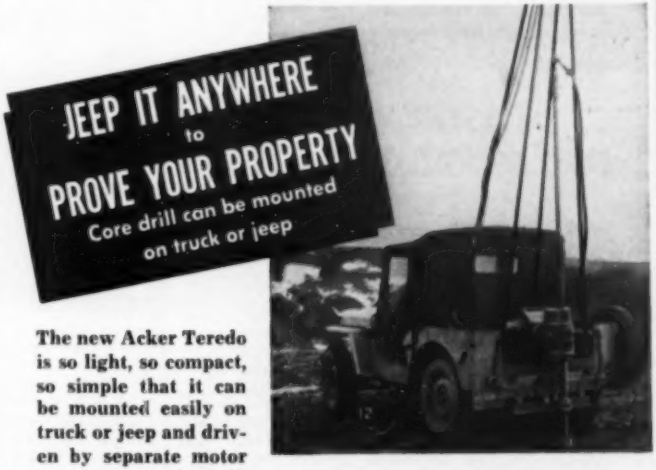
Donald C. Kimball, mining engineer, has been appointed superintendent of the Coons-Pacific Co. concentrating plant near Eveleth, Minn.

Milton H. Fies was named vice president in charge of mining for the Alabama Power Co. Mr. Fies has been consulting engineer for the company since 1937 and in charge of its coal mining operations since 1946.

A. F. HARPER

A. Finley Harper, formerly chief of the Safety Div., Birmingham, Alabama, State Department of Industrial Relations, has retired from active duty on account of ill health. Mr. Harper has been succeeded by **J. Gray Hanlin,** formerly state chief industrial inspector.

A. Roy Reed has accepted a position as mine superintendent for the Lepanto Consolidated Mining Co. He was formerly affiliated with the Eimco Corp.



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Personals, Continued

A. H. RICHARDS

A. H. Richards has again taken the position of consultant at the Longhorn Tin Smelter, Texas City, for the Reconstruction Finance Corp., after a two years retirement.

Henry E. Reed, Jr. has been appointed assistant geologist for the mineral development department of the Great Northern Ry. Co., Williston, N. D.

John A. Patterson is now project chief for the exploration branch, Atomic Energy Commission at Grand Junction, Colo.

David E. Lilienthal, formerly director of TVA, is now president of Minerals Separation North American Corp., Lakeland, Fla. Mr. Lilienthal succeeds **Dr. Seth Gregory,** who has been with the company for 25 years and who will continue as a consultant. **William Trotter,** after long service in the flotation field, has retired from active director of research, but is also continuing as a consultant. **Dr. Ernest W. Greene,** resident manager of the research laboratory, is now executive vice president and director of research; the new resident manager at Lakeland laboratory is **Wesley M. Houston,** formerly its chief engineer; **Earl H. Brown** has been appointed assistant to Dr. Greene, devoting a major portion of his time to iron ore; **Albert H. Pedler,** resident manager, is now in charge of the company's laboratory for iron ore flotation, Mesabi Range.

H. J. O'Carroll, formerly assistant smelter superintendent of Braden Copper Co. at Sewell, Chile, is now assistant general manager of Kennecott Copper Corp. at Ray, Arizona.

Charles A. Park of Benguet Consolidated Mining Co. has been transferred from its Masinloc, Zambales mine to its Antamok mine near Baguio.

Alan T. Broderick, formerly regional geologist with the M. A. Hanna Co., is now chief geologist with the Inland Steel Co.

DR. JOSEPH C. RINTELEN, JR.

Dr. Joseph C. Rintelen, Jr. has been named chairman of the Dept. of Mining and Metallurgy at Texas Western College to succeed Professor Graham. A member of the Texas Western College faculty since 1949 as an associate professor of metallurgy, Dr. Rintelen was promoted to the rank of full professor along with being named to head the Dept. of Mining and Metallurgy.

C. M. Dolliver, Jr. is now with the Summit Mining Corp. of Carlisle, Pa., as production manager developing a prophyllite deposit in Adams and Cumberland Counties.

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PROF. J. F. GRAHAM

Professor John Fraser Graham, a member of the Texas Western College faculty since 1925, is retiring as chairman of the Dept. of Mining and Metallurgy. Prior to joining the faculty of Texas Western, he served as professor of mining and metallurgy at New Mexico School of Mines and Wisconsin School of Mines. During his long career he worked with gold, copper, silver, iron, zinc, mercury, pyrite and antimony properties. Professor Graham has been active as a faculty sponsor of the student branch of the AIME and the Scientific Club, a social and service organization for engineering students. Professor Graham will continue as a part-time member of the faculty at Texas Western College.

Southall R. Pfund was elected vice president and counsel of the United States Smelitng, Refining & Mining Co. by the board of directors.

C. O. Perlwitz has accepted a position as construction engineer with Northern Peru Mining & Smelting Co. in their Shorey Unit.

Edward Prostel is employed as coal research engineer with the Natural Resources Research Institute of the University of Wyoming.

James B. Scott has resigned his position with the U. S. Geological Survey at Grand Junction, Colo., to become division geologist with the Oregon State Highway Dept. in Bend, Ore.

Roger Stern has been transferred from the Taxco Unit of American Smelting & Mining Co. where he was assistant mill superintendent to the Angangeo Unit as mill superintendent.

Robert G. Vervaeke has resigned as works manager of the Evans Plant of the United States Gypsum Co., and has accepted the position of vice president of operations of the Quincy Corp. at Quincy, Wash.

Gloyd M. Wiles has been named vice president and general manager of the Nickel Processing Corp.

James A. Barr, Jr., joined the Raw Materials Division of the Atomic Energy Commission on October 1. He will be working on the recovery of uranium from phosphate and minor sources. Mr. Barr leaves a position as sales manager of flotation chemicals for Armour & Co., Chemical Division, in Chicago to take up his new duties in Washington, D. C.

Professors **Vincent P. Gianella** and **Walter S. Palmer** who collectively gave 71 years of service to training mining students retired from the Mackay School of Mines, University of Nevada this summer.

As chairman of the Dept. of Geology-Geography, Dr. Gianella terminated 29 years of teaching at the Mackay School of Mines. A Californian by birth, Dr. Gianella received his early education in Marysville. He received his B.S. degrees in electrical engineering (1910) and mining engineering (1911) from Oregon State College. During his varied career Gianella worked in Alaska, Arizona, and California. Dr. Gianella received his Ph.D. in geology from Columbia in 1917. He earned a M.S. degree in mining engineering from University of Nevada and joined the faculty of the Mackay School of Mines in 1923.

Walter S. Palmer, E.M. professor of metallurgy and chairman of the Dept. of Metallurgy, gave 42 years of service to the University of Nevada. Born in Waterville, Maine, Mr. Palmer came to Reno in 1889 and received a B.S. degree in mining engineering from the University of Nevada in 1905. Before joining the staff at the University of Nevada, Walter Palmer gained practical experience in mining and metallurgy in Mexico, California, Pennsylvania and Nevada.

In 1916 he became chairman of the Dept. of Metallurgy at Nevada and in 1923 became director of the State Analytical Laboratory.

Appointed to the staff of the Mackay School of Mines to replace Professors Gianella and Palmer were **John S. Winston,** metallurgist, and **Lon S. McGirk, Jr.,** geologist.

Dr. McGirk, a graduate of Oregon State College and Stanford University has had professional experience in California, Oregon and Washington and with the Texas Gulf Sulphur Co. and the U. S. Geological Survey.

Professor Winston, a graduate of Cornell College, Iowa and the Missouri School of Mines, has taught metallurgy at Illinois Institute of Technology, the University of Chicago, the Biarritz American University, Valparaiso University, Morningside College and the Missouri School of Mines.

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George L. Richert (Member 1929), aged 54, died at his home in Oklahoma City May 26, 1952. In 1917 he enlisted in the Army while attending the Colorado School of Mines. After the Armistice he attended the college at Cleremot Ferrand, France, for one year, and then went to the Missouri School of Mines and graduated in 1922. The next 4 years were spent with the Kennecott Copper Corp. at Kennecott, Alaska. From 1926 to 1930 he was chief engineer at the Minas de Matahambre, Pinor del Rio, Cuba for the American Metals Co. He went to Ecuador for the South American Development Co. where he spent four years in Portorela and four years in Macushi. During World War II he served as special assistant to the Minerals Attache at the U. S. Embassy in Mexico City. For the past five years he has been in charge of the Engineering Dept. at the Dolese Bros. Co. in Oklahoma City. Mr. Richert is survived by his wife and daughter.

B. W. Jacobs (Member 1942) has died. At the time of his death he was an employee of International Minerals & Chemical Corp. Mr. Jacobs served in World War I as an electrician in the U. S. Navy. From 1920 to 1923 he was a mechanic in Tampa, Fla. Mr. Jacobs had been connected with Phosphate Mining Co., and International Agri Corp. He was refinery foreman for Union Potash and Chemical Co., in Carlsbad, N. Mex., and in 1941 was made refinery superintendent of this company. Born in Washington, Kansas, Mr. Jacobs was educated at Morris High School.

Thomas W. Page (Member 1944) died Sept. 5, 1952. A native of Charlottesville, Va., Mr. Page had been a resident of Middletown for 10 years. He was a member of the Sportsmen's Club and Sigma Rho fraternity. Mr. Page traveled extensively in his work for Armco, assisting in the development of iron ore, limestone and coal properties for the steel corporation. He joined the company in March, 1943, and assisted vice president L. F. Reinartz in the company's mining developments. During World War II, Mr. Page served for 2½ years with the steel division of the War Production Board in Washington. A graduate of the Montana School of Mines in 1934, he formerly was a general superintendent of several Canadian mines. Surviving are his wife, Susan; one son, Thomas W. Page IV; two daughters, Celeste and Virginia.

Dr. H. Ries (Member 1895) died in April, 1951. A native of Brooklyn, he attended Columbia University where he received his Ph.B., M.A., and Ph.D. degrees. An instructor at Cornell University from 1898 until his retirement in 1939, he was head of the geology department from 1914 to 1937. A past director of American Foundrymen's Association (1933 to 1936), Dr. Ries also served as technical director of the association's committee on molding sand research, chairman of the sand research project and head of many groups involved in that study. In 1936 he was awarded the Joseph S. Seaman Gold Medal. Dr. Ries was a member of the Canadian Mining Institute, American Ceramic Society (of which he was past president), Geologic Society of America and a fellow of the English Ceramic Society.

Harold A. McAllister (Member 1944) died. Born at Sunbury, Ohio and educated at Ohio State University, Mr. McAllister's first job was with the University of North Dakota as an instructor of mining and metallurgy. During his career in mining and coal preparation he worked for Elk Horn Mining Corp. and for Lorain Coal & Dock Co. as mining engineer. He was manager of operations for West Virginia Coal & Coke Co., and for Nutchinson Coal Co. Mr. McAllister was president of McCall Coal Co. and superintendent of Merrill Coal Mines. One of his last positions was with the Clean Eagle Coal Co. as vice president and general manager. Mr. McAllister graduated from Ohio State University with an E.M. degree.

J. G. Bently (Member 1934) died recently. Mr. Bently had a varied career in coal mining. He worked for the Nebraska Power Co., as operating engineer and was connected with the Peoples Coal Co. as combustion engineer in Omaha, Neb. For a few years he worked for Burson Knitting Co. in Rockford, Ill., as superintendend of maintenance. The Peabody Coal Co. of Chicago employed him as combustion engineer. In his position as fuel engineer with Sahara Coal Co. he did work in cooperation with production department tying preparation in with utilization. Mr. Bently was born in Colorado Springs, Colo.

Reginald K. Bailey (Member 1947) died recently. Mr. Bailey had been connected with the U. S. Geological Survey since 1912. His work was largely connected with discovery and development of potash in U. S. He held patents on the oxidation of methane and treatment of polyhalite to recovery of potash salts. Mr. Bailey discovered the first U. S. occurrences of several minerals and did work on minerals associated with potash. Born in Lawrence, Kansas, he was educated at the University of Kansas.

Appreciation of

Calm Morrison Hoke

by Florence E. Wall, FAIC

Calm Morrison Hoke, chemist and engineer, authority on precious metals, died at her home in Palisade, N. J., on July 13, 1952, of an embolism following a long illness with phlebitis.

Miss Hoke, a member of AIME since 1929, was born in Chicago, Aug. 25, 1887, the daughter of Samuel W. Hoke, a pioneer in the technology of platinum, and Utopia (Wright) Hoke. She was educated in the Chicago schools, started at Wittenberg College (Ohio), and received her A.B. in chemistry at Hunter College, New York in 1908. She returned to Chicago, took a B.S. there in biology, in 1910, and was continuing in graduate study when her father's purchase of a blow torch business plunged her into metallurgy.

It may be all very well for present-day employment counselors to say, "Solve your shortage of engineers by employing women." Calm Hoke used to say that when in her efforts to bring herself abreast of the new business, she tried to enter certain classes in engineering at Columbia, it was like blasting at a solid wall with her own blow torches. The principal objection to admitting her was that there never had been any women in those classes. As a compromise, she was

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admitted to the lectures in mining engineering, but not to the laboratory. However, she did receive her Master's degree in engineering—the first granted there to a woman—in 1913. Except for two seasons during World War I when she also taught chemistry at Columbia, Miss Hoke's entire professional career was devoted to business.

This was Hoke, Inc., and the associated Jewelers Technical Advice, established by her father in 1912. They soon held many patents for her inventions — small torches and burners, suitable for the delicate work of jewelers, which utilized ordinary illuminating gas instead of acetylene. She devised methods of extending the old touchstone method of testing gold and silver to the metals of the platinum group, then coming into use, and soon became an authority on the reclamation of precious metal wastes. She was vice-president and chief consultant of Hoke, Inc., from 1926 until the business was sold in 1934.

Through Jewelers Technical Advice, C. M. Hoke instructed jewelers, dentists, refiners, and many others in the melting, refining, salvaging, and finishing of all the precious metals. She gave many talks on the occupational opportunities in her unusual field, especially for girls in what she called gemology and metal book-keeping. She designed many beautiful pieces of jewelry, especially in platinum when it was very new. She had many calls for articles in the trade and professional journals; and her two books, *Testing Precious Metals* (1932) and *Refining Precious Metal Wastes* (1940) are classics in the industry.

Miss Hoke was a Fellow (and founder) of the American Institute of Chemists, and a Fellow of the AAAS; also a member of the American Chemical Society, the Electrochemical Society, the University of Chicago Alumni, Women's University Club (N. Y.), and several other organizations that represented her wide diversity of interests. She will long be remembered by all that knew her in any capacity, professional or personal, because of her exceptional accomplishments, her engaging personality, and the generous understanding she gave to all their problems.

Although always known simply as C. M. Hoke, she was married in 1927 to T. Robert McDearman, and is survived by her husband, a civil engineer, now with the N. J. State Highway Dept.

Ralph H. Sweetser (Member 1894) died on June 29. Mr. Sweetser was born in Portsmouth, N. H. on Mar. 7, 1871. He attended Massachusetts Institute of Technology and graduated with the degree of B.S. in mining engineering and metallurgy. In 1892 he joined the Maryland Steel Co., Sparrows Point, Md., working at the blast furnaces. In 1897 he was superintendent of the Earlston furnace, Everett, Pa. He joined the Salem Iron Co. as assistant to the president. Mr. Sweetser joined the Cranberry Furnace Co., John-City, Tenn. in 1902 as manager. He went to Canada a few years later and joined the Algoma Steel Co., Sault Ste. Marie, Ont. Returning to the United States in 1907, he accepted a position with the Columbus Iron & Steel Co., Columbus, Ohio. For three years he was president and general manager of the Thomas Iron Co., Easton, Pa. In 1917 Mr. Sweetser joined the American Rolling Mill Co., Columbus as works manager. He was appointed assistant

R. H. SWEETSER

(Continued on page 1097)

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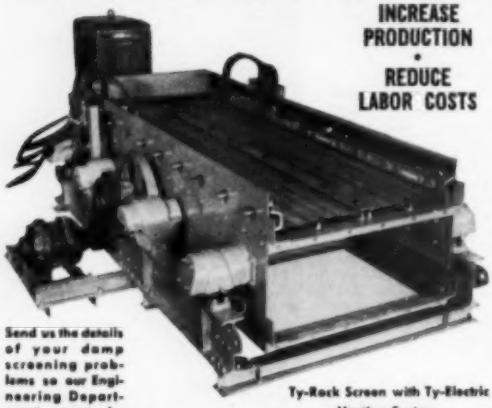

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to the first vice-president and was also vice-president of the Portsmouth By-Product Coke Co. In 1930 he was appointed assistant to vice-chairman, Armco. A Director and vice-president of AIME from 1925 to 1927, Mr. Sweetser was first Chairman of the Iron and Steel Div., and was Chairman of the Committee on Blast Furnace and Raw Materials. He was also a member of the American Iron and Steel Institute. In 1938 he wrote the book *Blast Furnace Practice*. He had been president of the Southern Ohio Pig Iron & Coke Assn. From 1930 until 1948 Mr. Sweetser was a consulting engineer practicing in New York.

Thomas H. Clagett (Member 1912) died in July 1952. He graduated with B.S. from Lehigh University. Mr. Clagett during his mining career worked for Michigan Coal Co., Jamestown Coal & Transportation Co., and Pocahontas Coal & Coke Co. His experience included work in construction (building), drafting, and surveying. At the time of his death he was vice president of Pocahontas Land Corp. Besides holding membership in AIME, he was a member of West Virginia Coal Mining Institute.

NECROLOGY

Elected Date	Name	Date of Death
1943	William W. Avery	June 13, 1952
1926	Charles E. Basso	July 17, 1952
1936	Henry M. Dawes	Sept. 29, 1952
1929	C. M. Hoke	July 13, 1952
1944	T. W. Page	Sept. 6, 1952
1905	John J. Rutledge	Sept. 13, 1952
1885	Arthur L. Walker	Sept. 30, 1952

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Total AIME membership on Sept. 30, 1952 was 18,226; in addition 1,561 Student Associates were enrolled.

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The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

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Superior—Brown, Sherburne (C/S—S-J)

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Little Rock—Randall, Bruce A. (J)

California
Darwin—Cunningham, Andrew B. (J)
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Coming Events

Nov. 4, 1952, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Nov. 5, AIME, Chicago Section, Chicago Bar Assn., Chicago.

Nov. 6-8, New Mexico Mining Assn. and International Mining Days, joint convention, Alvarado Hotel, El Paso.

Nov. 7, AIME, Pittsburgh Section, National Open Hearth Steel Committee, off-the-record meeting, Hotel William Penn, Pittsburgh.

Nov. 7-8, Birmingham Technical Meeting of Southeast Section, AIME, Birmingham.

Nov. 10-11, Eighth Annual Meeting of The Magnesium Assn., Hotel Biltmore, New York City.

Nov. 12, AIME, Connecticut Section, Hartford, Conn.

Nov. 18, AIME, Buffalo Section, National Open Hearth Steel Committee, Hotel Statler, Buffalo.

Nov. 19, AIME, Connecticut Section, Hartford, Conn.

Nov. 19, American Mining Congress Coal Div. Conference, Wm. Penn Hotel, Pittsburgh.

Nov. 19, Institute of Metals, lecture theater, Royal Institution of Great Britain, London.

Nov. 20-21, American Society for Quality Control, mid-west conference, Claypool Hotel, Indianapolis.

Dec. 2, American Mining Congress, annual membership meeting, University Club, New York.

Dec. 3, Society for Applied Spectroscopy, Socony-Vacuum Training Center, New York City.

Dec. 4-6, AIME, Electric Steel Furnace Conference, Hotel William Penn, Pittsburgh.

Dec. 5, Atlanta Meeting of Southeast Section, AIME, Ansley Hotel, Atlanta.

Dec. 7-10, American Institute of Chemical Engineers, annual meeting, Hotels Cleveland and Carter, Cleveland.

Dec. 8, AIME, Arizona Section, all-day meeting, Tucson.

Jan. 14, 1953, AIME Connecticut Section, Hammond Metallurgical Laboratory, Yale University, New Haven, Conn.

Jan. 21-23, Ninth Annual Technical Conference of Society of Plastics Engineers, Inc., Hotel Statler, Boston.

Jan. 26-30, American Society of Heating and Ventilating Engineers, international heating and ventilation exposition, International Amphitheatre, Chicago.

Feb. 16-19, AIME, annual meeting, Statler Hotel, Los Angeles.

Mar. 16-20, National Assn. of Corrosion Engineers, annual conference and exhibition, Hotel Sherman, Chicago.

Mar. 23-27, ASM western metal congress and exposition, Pan-Pacific Auditorium, Los Angeles.

Apr. 13-May 23, Empire Mining and Metallurgical Congress, Australia-New Zealand.

Apr. 20-22, AIME, National Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Hotel Statler, Buffalo.

May 14-16, Pacific-Northwest Metals and Minerals Conference of 1953, joint meeting of Metals Branch and Industrial Minerals Div., Ben Franklin Hotel, Seattle.

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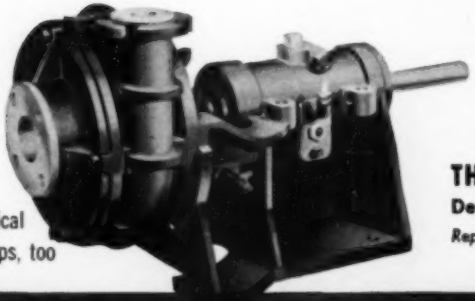

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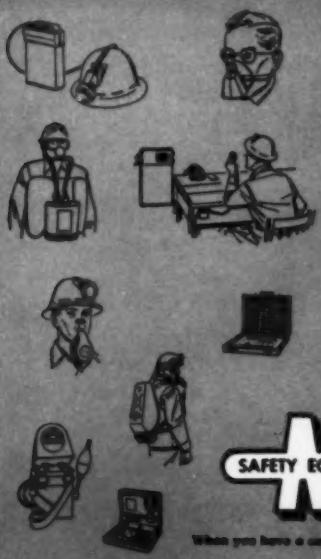

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